NOTE

This Microfiche meets the quality requirements of the American National Standard for Microfiche of Documents (ANSI PH5.9-1975). However, because of imperfections in the document from which it was made, it contains images that are illegible or difficult to read. The source document for this microfiche is the best copy that could be obtained prior to filming.

CIS

Congressional Information Service, Inc
Washington, D.C

STRATEGIC AND CRITICAL MINERAL POSITION OF THE UNITED STATES WITH RESPECT TO CHROMIUM, NICKEL, COBALT, MANGANESE, AND PLATINUM

A. Silverman
J. Schmidt
P. Queneau
W. Peters

DRAFT FINAL REPORT
June 15, 1983

"These Working Papers were prepared by outside contractors as an input to an ongoing OTA Assessment. They do not necessarily reflect the analytical findings of OTA. These Working Papers have been neither reviewed nor approved by the Technology Assessment Board."

J952-52

STRATEGIC AND CRITICAL MINERAL POSITION OF
THE UNITED STATES WITH RESPECT TO
CHROMIUM, NICKEL, COBALT, MANGANESE, AND PLATINUM

"This is an OTA Working Paper that was used in preparing OTA's final Assessment Report on Strategic Materials: Technologies to Reduce U.S. Import Vulnerability. OTA makes these Working Papers available for the use of readers desiring a more detailed or technical discussion of an issue than can normally be accommodated in our final Report. As an OTA Working Paper, this report has not been reviewed or approved by the Technology Assessment Board. The findings and conclusions expressed in the report are those of the authors and do not necessarily reflect the views of OTA, the Strategic Materials Advisory Panel, or the Technology Assessment Board."

TABLE OF CONTENTS

Page

CHAPTER I

INTRODUCTION.. 1

MINERAL RESERVES AND RESOURCES.. 2

U.S. RESOURCE BASE.. 3

STUDY FINDINGS.. 5

U.S. Metals Production in 1995.. 6

U.S. Metals Production in 2010.. 10

CHAPTER II DESCRIPTION AND USES OF METALS OF INTEREST

COBALT.. 16

Description and Uses.. 16

Types of Cobalt-Bearing Deposits...................................... 16

Stratabound Copper/Cobalt Deposits.................................... 17

Laterite Deposits... 18

Magmatic Sulfide.. 19

Hydrothermal Deposits... 21

Nickel Description and Uses, Types of Deposits........................ 22

Strataform/Massive Sulfides... 23

Laterite Deposits... 24

CHROMIUM.. 25

Description and Uses.. 25

Types of Deposits... 26

Strataform/Layered Deposits... 26

Podiform Deposits... 27

Laterite Deposits... 28

MANGANESE.. 28

Description and Uses.. 28

Types of Manganese-Bearing Deposits.............................. 29

Sedimentary Deposits.. 29

Hypogene or Exhalative Deposits................................ 31

Metamorphic Deposits.. 31

Surficial/Residual Deposits.................................... 32

PLATINUM... 32

Description and Uses.. 32

Types of Platinum-Bearing Deposits.............................. 33

CHAPTER II ESTIMATES OF THE METAL RESOURCE BASE........................ 35

CHROMIUM... 36

NICKEL... 37

COBALT... 40

MANGANESE.. 42

PLATINUM... 44

CHAPTER IV PRODUCTION HISTORY AND REAL PRICE OF METALS.................. 47

COBALT... 47

NICKEL... 49

CHROMITE... 51

MANGANESE.. 52

PLATINUM... 54

CHAPTER V CURRENT MINING ACTIVITIES AND PROPOSALS AND EVALUATION
OF ISSUES AFFECTING DEVELOPMENT...................................... 56

SUMMARY OF STATUS OF VARIOUS PROJECTS.................................. 56

SUMMARY OF ENVIRONMENTAL CONCERNS AT EXISTING AND PROPOSED PROJECTS.. 58

ENVIRONMENTAL CONSIDERATIONS AT OTHER DEPOSITS.......................... 59

GOODNEWS BAY.. 61

NICKEL MOUNTAIN.. 64

  Geology.. 64

  Mining... 66

BLACKBIRD PROJECT.. 67

  Environmental Setting.. 68

  Geology.. 70

  Historical Mining Activities and Current Development................. 71

NORANDA PROPOSAL... 74

CITIZEN CONCERNS AND OBJECTIVES.. 77

RIVER OF NO RETURN WILDERNESS... 78

GASQUET MOUNTAIN... 80

  Environmental Setting.. 80

  Socioeconomic Setting.. 84

  Geology.. 85

  Mining History in Area... 86

  Califonria's Nickel Proposal... 87

  Mining Method.. 88

  Processing Facilities.. 89

  Regulatory, Reclamation, and Environmental Issues Related to the Proposed Project.. 90

  Other Concerns... 94

STILLWATER COMPLEX/BEARTOOTH FACE.......... 95

Environmental Setting.......... 96

Sociology.......... 97

Geology.......... 98

History of Mining Development in the Stillwater Complex.......... 99

ANACONDA MINERALS-STILLWATER PROJECT PROPOSAL.......... 103

Citizen Concerns about Mining Development.......... 105

MADISON MINE/SOUTHEASTERN MISSOURI.......... 108

DULUTH COMPLEX/NORTHEASTERN MINNESOTA.......... 111

Environmental Setting.......... 112

Geology.......... 114

Mining Industry Interest.......... 116

Environmental Impacts of Development.......... 118

REFERENCES.......... 123

CHAPTER VI PROCESS TECHNOLOGY FOR DOMESTIC SOURCES OF CHROMIUM, COBALT, MANGANESE, NICKEL, AND PLATINUM CHROMIUM 126

Introduction.......... 126

Conclusions.......... 128

Stillwater Chromite.......... 128

California-Oregon Disseminated Chromite.......... 131

Chromite from Nickel-Cobalt Laterite.......... 133

Production of Ferrochromium.......... 135

Prereduction Followed by Leaching.......... 135

Conventional Electric Furnace Operation.......... 136

Recent Smelting Innovations.......... 136

Thermal-Plasma Smelting.......... 138

Pamco Hot-Alloy Smelting 139

COBALT 140

Introduction 140

Conclusions 142

Idaho's Blackbird Mine 143

Cobalt Concentrate Refining 146

The Madison Mine, Missouri 149

Co-Ni Concentrate Refining 150

Cobalt Recovery from Missouri Lead Operations 155

MANGANESE 158

Conclusions and Recommendations 158

Historical Domestic Production 160

Physical Beneficiation of Manganese Ores 160

Processing of Cuyuna Manganese Concentrates 163

Lead Time for Domestic Manganese Production 164

Solution Mining of Manganese Ores 165

Biochemical Leaching and Molecular Biology 167

Feedstock Considerations 167

Reactor Constraints 167

Removal of Reaction Products 168

Conclusion 168

A Mexican Connection 169

Stan Margolin's ADL Contract 170

NICKEL 171

Introduction 171

Conclusions 172

Minnesota's Duluth Complex 173

Smelting Duluth Complex Concentrates 178
Recovery of Cobalt 180
General Observations 181
Hanna's Nickel Mountain Operation 182
Smelting to Ferronickel 183
Ni-Co from Nickeliferous Laterite by Hydrometallurgy 186
PLATINUM 188
Introduction 188
Conclusion 189
Montana's Stillwater Complex 190
Beneficiation 190
Smelting 192
Minnesota's Duluth Complex 196
Alaska's Goodnews Bay 197
BIBLIOGRAPHY 200

CHAPTER VII POTENTIAL INNOVATIONS IN MINING AND METALLURGY APPLIED TO DOMESTIC RESOURCES OF CHROMIUM, COBALT.NICKEL, MANGANESE, AND PLATINUM, 1990-1992 AND 1995-2010 209
HORIZONS IN MINING 210
Premining Investigations 211
Open-Pit Mining 212
Underground Mining 213
In-Situ Mining 216
MINING APPLICATION TO CHROMIUM, COBALT, NICKEL, MANGANESE, AND PLATINUM DEPOSITS 218
Chromite Deposits 218

Cobalt Deposits.. 219

Nickel Deposits.. 219

Manganese Deposits... 220

Platinum Deposits.. 220

HORIZONS IN METALLURGY.. 220

Mineral Processing... 221

Extraction and Refining.. 223

MINING APPLICATION TO CHROMIUM, COBALT, NICKEL, MANGANESE, AND.
PLATINUM DEPOSITS.. 223

Chromite... 223

Cobalt... 224

Nickel... 225

Manganese.. 225

Platinum... 226

CONCLUSIONS.. 226

1990-1992.. 227

1995-2010.. 228

Economic Considerations.. 228

BIBLIOGRAPHY... 230

CHAPTER VIII U.S. METALS PRODUCTION IN THE YEAR 1995..................... 232

CHROMITE ANALYSIS.. 232

NICKEL ANALYSIS.. 237

COBALT ANALYSIS.. 243

MANGANESE ANALYSIS... 250

PLATINUM ANALYSIS.. 256

CHAPTER IX THE STRATEGIC MINERAL DEVELOPMENT IN THE U.S. FOR THE YEAR 2010 261
INTRODUCTION.......... 261
CHROMITE IN THE YEAR 2010.......... 262
NICKEL IN THE YEAR 2010.......... 265
COBALT IN THE YEAR 2010.......... 269
PLATINUM IN THE YEAR 2010.......... 271
MANGANESE IN THE YEAR 2010.......... 274
SUMMARY AND CONCLUSIONS.......... 276

LIST OF TABLES

Following
Page

Chapter 1

Table 1-1 Nickel Reserves and Resources.............................6
Table 1-2 Cobalt Reserves and Resources.............................6
Table 1-3 Chromium Reserves and Resources...........................6
Table 1-4 Manganese Reserves and Resources..........................6
Table 1-5 Platinum Group Metals Reserves and Resources..............6
Table 1-6 Principal United States Nickel Deposits...................6
Table 1-7 Principal United States Manganese Deposits................6
Table 1-8 Principal United States Chromium Deposits.................6
Table 1-9 Principal United States Platinum Group Metals Deposits.....6
Table 1-10 Principal United States Cobalt Deposits...................6

Chapter 2

Table 2-1 World Cobalt Production and Reserves by Geologic Type.... 17
Table 2-2 Cobalt Resources Estimate.................................. 19
Table 2-3 Major Cobalt Minerals of Economic Importance............. 19
Table 2-4 Principle Nickel Minerals.................................. 22
Table 2-5 U.S. Bureau of Mines and United National Estimates of World Reserves and Resources of Nickel, 1979........... 23
Table 2-6 Characteristics of the Major Known Stratiform Chromite Deposits.. 26
Table 2-7 Estimated World Chromium Resources by Deposit Type and Product Grade.. 27
Table 2-8 World Manganese Ore Resources............................ 29
Table 2-9 Identified, Hypothetical, and Speculative Resources of Platinum-Group Metals.............................. 33

Chapter 3

Table 3-1 Estimated World Chromium Reserves and Resources by Country and Region.................................. 36
Table 3-2 Estimates of World Cobalt Reserves and Resources......... 40
Table 3-3A&B World Mine Production and Reserve Base of Manganese...... 42
Table 3-4 Major Identified Manganese Resources of the United States 43
Table 3-5 World Mine Production and Reserve Base of Platinum....... 44

Chapter 4

Table 4-1 Mine Production of Cobalt in the Noncommunist World...... 47
Table 4-2 Projected Production of Extractable Cobalt............... 48
Table 4-3 Time, Producer Price Relationship for Cobalt (1957-1980). 48
Table 4-4 Breakdown of Cobalt Production by State and Region....... 49
Table 4-5 World Nickel Production by Country, 1973-1979............ 50
Table 4-6 Projected Mine Nickel Production Capacity................ 51

Table 4-7 Time, Producer Price Relationship for Nickel, 1955-1980.. 51
Table 4-8 State and Region Brakdown of Nickel...................... 51
Table 4-9 World Mine Production of Chromite by Country, 1974-1979.. 51
Table 4-10 United States Import Sources for Chromite................ 52
Table 4-11 United States Import Sources for Ferrochromium........... 52
Table 4-12 Time, Value Relationship for Chromium.................... 52
Table 4-13 Breakdown of Chromium Production by State and County..... 52
Table 4-14 World Apparent Consumption of Manganese Ore, 1960-1977, by Country.................................. 53
Table 4-15 World Apparent Consumption of Ferromanganese, 1960-1977, by Country.................................. 53
Table 4-16 Projections of World Manganese Mining Capacity, 1980-2000, Compared with 1975 Capacity and with Estimated and Projected Manganese Ore Consumption...... 53
Table 4-17 Trends in Manganese Prices and Costs, 1970-2000.......... 53
Table 4-18 Projections of World Ferromanganese Capacity, 1980-200054 Compared with 1986 Capacity and with Estimated and Projected Ferromanganese Consumption.................. 54
Table 4-19 State Breakdown for Manganese Production................. 54
Table 4-20 Breakdown of Platinum Production by State and Region..... 55

Chapter 5

Table 5-1a Summary of Regulatory and Environmental Issues at Likely Projects... 59
Table 5-1b Long-term Speculative Projects........................... 60
Table 5-2 Classification Characteristics of Ore Types at Nickel Mountain Mine................................... 65
Table 5-3 Gasquet Mountain Project Ore Reserves.................... 87
Table 5-4 Copper Ore Tonnage, Grade, and Metal Content Estimates by Resource Zone...................................... 115
Table 5-5 Environmental Considerations and Mineral Resources--Duluth Complex... 121

Chapter 8

Table 8-1 Chromium Information...................................... 232
Table 8-2 Chromite Resources.. 232
Table 8-3 Chromium-Laterite Deposit Resources....................... 232
Table 8-4 Chromium Costs.. 232
Table 8-5 United States Net Import Position for Chromite and Ferrochromium.. 233
Table 8-6 Chromium Cost Analysis by Deposit Type.................... 233
Table 8-7 Nickel Information.. 237
Table 8-8 Nickel Resources (by Deposit Size)........................ 237
Table 8-9 Nickel Resources (by Deposit Type)........................ 237
Table 8-10 Nickel Costs (by Deposit Size)........................... 237
Table 8-11 Nickel Costs (by Deposit Type)........................... 237
Table 8-12 Nickel Cost Analysis by Type Geologic Deposit............ 239
Table 8-13 Nickel Cost Analysis by Size Deposit..................... 239
Table 8-14 Potential United States Cobalt Production................ 243
Table 8-15 Cobalt Information....................................... 244

Table 8-16 Cobalt Resources (by Deposit Size)........................ 244
Table 8-17 Cobalt Resources (by Deposit Type)........................ 244
Table 8-18 Cobalt Costs (by Deposit Type)............................ 244
Table 8-19 Cobalt Cost Analysis...................................... 244
Table 8-20 Manganese Information..................................... 251
Table 8-21 Manganese Resources....................................... 251
Table 8-22 U.S. Bureau of Mines Manganese Cost Analysis.............. 251
Table 8-23 Manganese Mine-Life Costs................................. 256
Table 8-24 Platinum Resources.. 256

LIST OF FIGURES

Following Page

Chapter 2

Figure 2-1 Important Nickel Reserves and Resources.................. 23
Figure 2-2 Important Chromite Reserves and Resources in Stratiform Deposits.................................. 26
Figure 2-3 Important Chromite Reserves and Resources in Podiform Deposits..................................... 27

Chapter 4

Figure 4-1 Cobalt: Total U.S. Production........................... 49
Figure 4-2 Nickel: Total U.S. Production........................... 49
Figure 4-3 Chrome: Total U.S. Production........................... 52
Figure 4-4 Manganese: Total U.S. Production........................ 52
Figure 4-5 Platinum: Total U.S. Production......................... 55

Chapter 5

Figure 5-1 Geologic Map and Cross-section of Nickel Mountain Deposit Showing Upper Peridotite Thrust Plate and Outlines of Orebodies................................. 65
Figure 5-2 Location of Blackbird Mine............................... 67
Figure 5-3 Spatial Changes in pH and Copper Concentration in Panther Creek.. 69
Figure 5-4 Longitudinal Projection of Blackbird Mine and Plan of 6850 Level, Blackbird Mine, Lemhi County, Idaho........ 70
Figure 5-5 Plan View and Longitudinal Projection of the Blackbird Mine... 73
Figure 5-6 Flow Diagram of Flotation Process........................ 76
Figure 5-7 Special Mining Management Zone........................... 78
Figure 5-8 Regional Location of California Nickel Corp. Mine........ 80
Figure 5-9 Drainage Basins and Surface Water Sampling Stations...... 81
Figure 5-10 Bedrock Types.. 85
Figure 5-11 Typical Profile through Nickeliferous Laterite Deposits.. 86
Figure 5-12 Mining Sequence.. 88
Figure 5-13 Backfilling Sequence..................................... 88
Figure 5-14 Generalized Geologic Map of the Stillwater Complex....... 99
Figure 5-15 Mineral Ownership of the Stillwater Complex.............. 101
Figure 5-16 Topography of the Permit Area........................... 103
Figure 5-17 Artist's View of Stillwater River Valley, Showing Project Area... 103
Figure 5-18 Simplified View of Mining Method........................ 103
Figure 5-19 The Southeast Missouri Lead-Zinc-Copper Mining District.. 108
Figure 5-20 Schematic Mining Section................................ 109
Figure 5-21 Proposed Surface Plan................................... 110
Figure 5-22 Generalized Geologic Cross-Section...................... 110
Figure 5-23 MEQB Regional Copper-Nickel Study Mineralization Location Map.. 112
Figure 5-24 Cross-Section... 115

Figure 5-25 Blasthole Open Stoping.................................. 117
Figure 5-26 Smelter Siting Zones Surrounding Class PSD Areas......... 118
Figure 5-27 Potential of Development Zones Outside the B.W.C.A....... 120

Chapter 6

Figure 6-1 Size Range Applicability of Commercial Unit Processes.... 132
Figure 6-2 Schematic Cross-Section of a Submerged-Arc Furnace Used to Produce High-Carbon Ferrochromium.................. 136
Figure 6-3 Thermal Plasma Furnace.................................. 138
Figure 6-4 Stainless Steel Production at the Hachinoke Works of the Pacific Metals Co. Using the Pamco Hot-Alloy Process... 139
Figure 6-5 Blackbird Pilot Plant.................................. 144
Figure 6-6 Flow Diagram for the Treatment of Blackbird Bulk Concentrate... 144
Figure 6-7 Flow Diagram of Anschutz's Proposed Mine-Run Ore Beneficiation Plant.................................. 149
Figure 6-8 Flow Diagram for Anschutz's Proposed Tailing-Ore Beneficiation System................................. 150
Figure 6-9 Metallurgical Balance: Cobalt........................... 150
Figure 6-10 Metallurgical Balance: Nickel........................... 150
Figure 6-11 Metallurgical Balance: Copper........................... 150
Figure 6-12 Distributions of Siegenite When Processing Missouri Lead Ore... 155
Figure 6-13 Time and Cost to Commercialize New Process When Semi-Works Plant is Required.......................... 165
Figure 6-14 Time and Cost to Commercialize New Process When Semi-Works Plant is Not Required...................... 165
Figure 6-15 Bulk Flotation of Minnesota Duluth Gabbro Ore............ 174
Figure 6-16 Differential Flotation of Duluth Gabbro Bulk Concentrate. 174
Figure 6-17 Alternative Flow Diagrams for Smelting Duluth Gabbro ~~Bulk~~ Flotation Concentrate............................ 181
Figure 6-18 Production of Ferronickel at Hanna's Operation in Riddle, Oregon... 183
Figure 6-19 Nicaro Operation for Selective Reduction of Laterite Ore. 186
Figure 6-20 Leaching Selectively Reduced Laterite at Nicaro.......... 186
Figure 6-21 Moa Bay Flowsheet for Acid Pressure Leaching of Laterite Ore, as Improved by Amax, Inc.......................... 186
Figure 6-22 Impala Platinum's Concentrator Flowsheet................. 190
Figure 6-23 Impala Platinum's Smelter Flowsheet...................... 192
Figure 6-24 INCO's Copper Cliff Refining Complex..................... 192
Figure 6-25 Recovery of Nickel and Copper from High-Grade Matte...... 192
Figure 6-26 Schematic Flowsheet of the Falconbridge Nikkelverk A/S Plant for Treating High-grade PGM Matte................ 193
Figure 6-27 INCO's Solvent-Extraction Process for Separation and Purification of Precious Metals....................... 195

Chapter 8

Figure 8-1 Annual Production of Domestic Manganese for Selected Years at Various Prices............................... 251

Chapter I. SUMMARY AND CONCLUSIONS

Study Parameters

Since the end of World War II and the beginning of the cold war, the United States has had a public concern about the supply and availability of a number of minerals that are important, in some cases irreplaceable, in both strategic military and economic terms. At least four of these elements are the subject of this study, namely chromium, cobalt, nickel, and manganese. More recently, platinum has also become an element of concern primarily because of its increased use in catalytic converters and its restricted geologic availability.

The ferroalloy elements (chromium, nickel, cobalt, and manganese) show an overlapping and compatible system of geologic occurrence, economic vitality, major potential producers, end-use products, and import dependence that allows some generalization. It is important to remember, however, that each of these commodities has independent geologic and economic settings that must be evaluated separately to provide appropriate analysis for long-term availability and potential U.S. mineral resource vulnerability. This real or perceived vulnerability is, in fact, the focus of this study, which emphasizes the possible production and processing of domestic deposits of chromium, cobalt, manganese, nickel, and platinum.

This report not only evaluates the possibilities for domestic production of deposits of interest. It seeks to analyze the characteristics of world and U.S. deposits of interest, the realistic chances of production from U.S. resources, and the conditions under which such production could occur.

Reasonable expectations about the international and national structure of the ferroalloy industry and platinum can be made today on the basis of current knowledge and projected worldwide expectations over the next 25 to 30 years. Assessments for the year 2010 are, of course, very uncertain; however, this uncertainty can be made more manageable by considering individually the factors affecting it: (1) the demand for these commodities in the world and U.S.; (2) the prospects of new discovery and the rate of development of new-found resources; (3) the ability to develop innovative recovery technologies, thereby reducing the relative cost and price or increasing profit margins; (4) the ability to substitute for the ferroalloys, so they become less important in the total structure of the materials cycle 30 years from now than they are today; and (5) the geopolitical alignments that would allow new discoveries or known resources to compete in the international marketing system.

Mineral Reserves and Resources

The following discussion summarizes the geologic environment in which these five strategic materials are found, categorizes deposits according to geologic type following the guidelines of the U.S. Geological Survey, and looks at the size, grade, and distribution of these resources in the United States and abroad. In this discussion, the terms reserves and resources are differentiated in the following way. Most geologic and resource professionals classify deposits according to the system outlined by the U.S. Bureau of Mines and the U.S. Geological Survey (1976). This system defines reserves as "that portion of the identified resource from which a useable mineral or energy commodity can be economically and legally extracted at the time of determination." The term ore is used for reserves of some minerals. Resources, on the other hand, are that portion of the geologic endowment from which mineral or energy materials cannot be recovered currently but may be in the foreseeable future as changes in geologic knowledge or the economic situation and/or technology make mining feasible. Reserve data for demonstrated deposits are generally characterized by the subcategories of measured, indicated and inferred ore, reflecting a decreasing degree of certainty about tonnage and grade. In contrast, resources may or may not be demonstrated and it is important to note that most geologic evaluations of resources fail to give probabilities for quoted grade and tonnage numbers. Undiscovered resources can be placed in one of two categories; hypothetical or speculative resources. Hypothetical resources are those that can be reasonably expected to exist in the extension of known or partially explored mining districts or in areas that are similar to known mining districts. Speculative resources are those deposits that might occur in as yet unexplored geologic settings, containing perhaps new deposit types as well as those types known from old districts.

Reserve and resource estimates are not static. They change with the level of exploration, the market price of the commodities, changes in environmental constraints on exploration, mining and processing technology, and changes in end-use. It is important in resource assessment and policy planning and implementation to suggest the limits of reserve and resource data through probability estimates wherever and whenever appropriate. It is also important to suggest a time frame in which the reserve and resource base evaluations might become available to commerce through exploration and mining development.

U.S. Resource Base

Because the United States relies heavily on imports for these commodities of interest (Cr, Ni, Co, Mn, Pt) and current domestic production for all of these materials is practically nonexistent, the questions to be asked about vulnerability can be structured in the following way: first, is there a land-based continental resource for these minerals in the United States that is currently known, or that can be discovered with a reasonable exploration effort within a reasonably short time frame? Second, what is the relative economic availability of an identified potential supply for the five commodities of interest? Third, will the commodities be available to the U.S. in an amount, and over a sufficient time period, as to warrant significant private industry investment? Lastly, how does the U.S. supply compare to current and future worldwide availability in the time frame specified by this study (1995 and 2010)?

The U.S. is not now, nor has ever been, a major producer of the five commodities this study concerns. In addition the worldwide economic mineral distribution of these materials makes it unlikely that the U.S. will be a major supplier of any of these commodities in any near-term time frame, such as in the next 25 to 30 years. It is the occurrence, grade and size of deposits of these commodities in different geologic terrains that essentially fixes the domination of foreseeable world production to a few geologic environments, and to a small number of geographic areas. Unfortunately, the United States does not appear to have many suitable geologic systems of the grade and tonnage that are adequate to make them exploitable within the foreseeable future.

Manganese and chromium, and for the most part platinum, are derived from world-class deposits where they are the primary ore. The other metals of interest, cobalt and nickel, are both byproduct materials derived from mining ventures in which other metals are the essential economic concern, or the nickel and cobalt are the primary ores. There are a couple of examples of this primary ore in known U.S. deposits of cobalt and nickel.

Generally, the byproduct status for nickel and cobalt means that recoveries are tied to the production of some other major commodity; therefore, only a small incremental cost is usually needed to recover the byproduct and provide a new supply to commodity markets. The capacity to produce cobalt and nickel is primarily a function of the market outlook for other mineral commodities; on the other hand, due to the location of the important deposits in third world countries, their volume and pricing may be more a function of national cash flow requirements than profit-motivated private capital. Lastly, the size of the worldwide reserve and resource base for at least three of the five commodities of interest--nickel, chromium and manganese--is so large that marginal to submarginal deposits, including those of the United States, will be difficult if not impossible to integrate into domestic or world trade within the next 30 years, if free market conditions are assumed to exist.

Study Findings

The findings of this study have been organized as follows: Chapter II describes the metals and their uses; Chapter III then estimates the domestic resource base and places it, on the basis of size and grade of deposits, within the context of world class deposits. Chapter IV outlines the history of production of these materials. Chapter V focuses on those deposits with near-term development potential and discusses mining approaches and important environmental and socioeconomimc issues surrounding various development proposals. The discussion in Chapter VI turns to current processing options

Table 1. Nickel Reserves and Resources (000 short tons)

Country	RESERVES Ni Reserves*	Ore Grade (%)	Deposit Type	RESOURCES Ni Resources**	Ore Grade*** %	Deposit Type
United States	---	---	---	14,900	1.5-0.3	Laterite; nickel sulfide
New Caledonia	15,000	1-3	Laterite	31,000	1.8	Laterite
Philippines	5,700	0.4-2.2	Laterite	10,600	0.8	Laterite
Cuba	3,400	1.4	Laterite	14,200	1.0	Laterite
Canada	8,700	1.5-3	Nickel sulfide	12,500	1.0	Nickel sulfide
USSR	8,100	1.5	Laterite Nickel sulfide	13,200	0.6-1.5	Laterite; nickel sulfide
Indonesia	7,800	1.8	Laterite	55,000	0.9	Laterite
Australia	5,600	1.5-3	Laterite Nickel sulfide	3,200	1.2-1.5	Laterite; nickel sulfide
Zimbabwe	661 (UN)	---	---			
Greece	2,400 (UN)	---	---			
TOTAL****	57,361			154,600		

*Reserve figures and ore grade from USBM Mineral Commodity Summaries, 1983. USBM estimate of reserve base includes resources that are currently economic, marginally economic reserves and some that are currently subeconomic; therefore, figures may include nickel deposits usually categorized as resources rather than reserves.

**Mineral Facts and Problems, USBM, 1980. (Resources = specifically identified deposits whose contained minerals cannot be recovered with existing technology and economic conditions.)

***Professional Paper 820, USGS, 1973.

****Total represents major reserves and resources; other small miscellaneous deposits are not included.

Table 2. Cobalt Reserves and Resources (metric tons)

Country	Co Reserves*	RESERVES Ore Grade (%)**	Deposit Type**	Co Resource*	RESOURCES Ore Grade (%)	Deposit Type**
Philippines	181,000 (425,000)	.03-0.12	Laterite	23,000 (1,750,000)	.03-0.12	Laterite
Cuba	181,000 (800,000)	.03-0.12	Laterite	953,000 (300,000)	.03-0.12	Laterite
New Caledonia	91,000 (385,000)	.03-.05	Laterite	295,000 (450,000)	.03-0.05	Laterite
Indonesia	--- (565,000)		Laterite	--- (300,000)		Laterite
Zaire	1,179,000 (450,000)	0.1-0.45	Stratabound copper	227,000 (230,000)	0.1-0.45	Stratabound copper
Zambia	363,000 (300,000)	.09-0.4	Stratabound copper	227,000 (50,000)	.09-0.4	Stratabound copper
USSR	227,000 (200,000)	.03-0.1	Laterite Magmatic sulfide	23,000 (150,000)	.03-0.1	Laterite Magmatic sulfide
Australia	45,000 (135,000)	.08-0.5	Laterite Magmatic sulfide	249,000 (165,000)	.08-0.5	Laterite Magmatic sulfide
Canada	27,000 (30,000)	.03-0.11	Magmatic sulfide	227,000 (220,000)	.03-0.11	Magmatic sulfide
United States	---	---	---	770,000	.01-0.6	Laterite; magmatic sulfide
TOTAL***	2,294,000			2,994,000		

*Data from USBM, 1980, Mineral Facts and Problems. Figures in parentheses from Berndesanstalt fur Geowissenchaften und Rohstoffe (Hanover), 1978. (Definitions: Reserves = identified deposits from which minerals can be extracted with existing technology and under present economic conditions; conditional resources = specifically identified deposits whose contained minerals cannot be recovered with existing technology and economic conditions.

Ore grade and deposit type [illegible] (F[illegible] 1981). *[illegible] reserves and resources [illegible] small deposits not included.

Table 3. Chromium Reserves and Resources (000 long tons)*

Country	RESERVES Cr Reserves	 Ore Grade (%)	 Deposit Type	RESOURCES Cr Resources	 Ore Grade (%)	 Deposit Type
United States	---	---	---	350	46	Podshaped
Brazil	6,000	40-46	Stratiform	5,150	40-46	Stratiform
Cuba	250	46	Podshaped	1,100	46	Podshaped
India	7,000	40-46	Stratiform Podshaped	6,000	40-46	Stratiform Podshaped
USSR	21,000(?)	40-46	Podshaped Stratiform	22,000	40-46	Stratiform Podshaped
Finland	10,000	40-46	Stratiform	5,000	40-46	Stratiform
Turkey	5,000	46	Podshaped	5,000	46	Podshaped
Malagasy Republic	5,000	46	Stratiform(?)	6,000	40-46	Stratiform(?)
Zimbabwe	550,000	40-46	Stratiform Placer	500,000	40-46	Stratiform Placer
South Africa	1,050,000	40-46	Stratiform	2,050,000	40-46	Stratiform
Philippines	4,700	46	Podshaped	2,500	46	Podshaped
Iran	1,500	46	Podshaped	1,000	46	Podshaped
TOTAL**	1,660,450			2,604,100		

---

*Data from USGS Professional Paper 820, U.S. Mineral Resources, p. 117. (Reserves = identified deposits from which minerals can be extracted with existing technology and under present economic conditions; conditional resources = specifically identified deposits whose contained minerals cannot be recovered with existing technology and economic conditions.

**Total represents major reserves and resources; other small miscellaneous deposits not included.

Table 4. Manganese Reserves and Resources (000 metric tons)*

Country	RESERVES Mn Reserves	 Ore Grade (%)	 Deposit Type	RESOURCES Mn Resources	 Ore Grade (%)	 Deposit Type
South Africa	901,600	29-39	Mixed carb/oxide, oxide, carb (sed, surficial/resid)	1,960,000	27-36	Carb/oxides/silicate (sed, surf/resid)
USSR	2,278,800	18.5-25	Oxide, carb, mixed carb (sed, surficial/resid)	152,000	4-25	Mostly carb/silicate (sed/meta/resid)
United States	---	---	---	925,000	4-9	Carb/silicate, silicate/carb (sed/meta); oxides (sed?/hypogene); oxide/silicate/carb (sed/resid); oxide/carb (sed/meta/resid)
Gabon	194,000	50	Oxide (sed/surf-resid)	---	----	Carb/silicate (sed/meta)
Canada	---	---	---	193,000	9	Carb/silicate (sed/meta)
Australia	213,000	48	Oxide (sed)	100	52	Oxide (sed)
Brazil	60,609	43 (inferred only)	Oxide (lump), oxide (pellets); Carbonate (sed/meta, surf/resid)	69,200	20-47	Oxide (sed/resid)
India	53,650	38-48	Oxide, silicate (sed) Oxide (sed/meta)	---	----	----
Hungary	---	---	---	215,000	20.5	Carb (sed/exhal)
TOTAL	3,701,659			3,514,300		

---

*Data from NAS National Materials Advisory Board, 1981, Manganese Reserves and Resources of the World and their Industrial Implications. (Reserves = identified deposits from which minerals can be extracted with existing technology and under present economic conditions; resources = specifically identified deposits whose contained minerals cannot be recovered with existing technology and economic conditions.

**Total represents major reserves and resources; other small miscellaneous deposits not included.

Table 5. PGM Reserves and Resources

	RESERVES			RESOURCES		
Country	PGM Reserves (mil.tr. oz)*	Ore Grade (oz/ton)**	Deposit Type**	PGM Resources *** (mil.tr.oz)	Ore Grade	Deposit Type
South Africa	970	.09-.35	Stratiform	1,430	NA	Stratiform
USSR	200	.305	Stratiform, alpine placer, sulfide	200	NA	Stratiform, alpine, sulfide, placer
Canada	9	.026-.11	Stratiform	7	NA	Stratiform, sulfide
Colombia	<.1	---	Placer	4	NA	Placer
United States	16	.02-.45	Stratiform, placer	285	NA	Stratiform, placer, sulfide
Zimbabwe	--	---	---	100	NA	Stratiform
TOTAL	1,195			2,026		

*Data from USBM Mineral Commodity Profiles, 1983, Platinum-Group Metals. Estimate of reserve base includes resources that are currently economic, marginally economic reserves, and some that are currently subeconomic; therefore, figures may include nickel deposits usually categorized as resources rather than reserves.

**Data from USGS Professional Paper 820, 1973.

***Data from Mineral Facts and Problems, USBM, 1980. Resources = specifically identified deposits whose contained minerals cannot be recovered with existing technology and economic conditions.

Table 6. Principal United States Nickel Deposits

Deposit	Reserves (metric tons)	Est. Production Rate (metric tons/yr)	Est. Duration of Production (yrs)	Necessary Startup Time (yrs)	Est. Price for Mine to be Economic ($/lb)**
Nickel Mt. (Oregon)	100,000	5,000	20	0	3.02-4.55
Gasquet Mt. (Calif.)	157,909	8,773	21	7	3.02-4.55
Madison (Mo.)	7,000	700	10	3	12.51
Brady Glacier (Alaska)	128,953	11,723	11	9	4.89
Minnamax (Minn.)	480,643	11,723	41	3	4.89
TOTAL					

---

*Tons of recoverable Ni.
**Estimated price based on total production costs plus 20% return on investment.

Data compiled from Tables 8-7, 8-10, 8-11, 8-12, and 8-13. (Reserves = identified deposits from which minerals can be extracted with existing technology and under present economic conditions.)

The 1982 average producer price for nickel according to U.S. Bureau of Mines, Mineral Commodities Summaries 1983, was $3.20/lb.

Table 7. Principal United States Manganese Deposits

Deposit	Reserves (metric tons)	Est. Production Rate (metric tons/yr)*	Est. Duration of Production (yrs)	Necessary Startup Time (yrs)	Est. Price for Mine to be Economic ($/long ton unit unit)*
Maggie Mine (Ariz.)	300,000	30,000	10	4	10.44
Sunnyside (Colo.)	600,000	30,000	20	4	10.44
Maple/Hovey (Maine)	8,540,000	140,000	61	6	6.00
Cuyuna North Range (Minn.)	1,960,000	140,000	14	4	6.00
TOTAL	11,400,000				

*Tons of recoverable Mn.
**Estimated price based on total production costs per long ton unit plus 20% return on investment.

Data compiled from Tables 8-20 and 8-23. (Reserves = identified deposits from which minerals can be extracted with existing technology and under present economic conditions).

U.S. Bureau of Mines, Minerals Commodities Summaries 1983, reports a 1982 average import price range for 46-48% manganese ore of $1.58-1.68 per ton ton unit cont. mn. c.i.f., U.S. ports.

Table 8. Principal United States Chromium Deposits

Deposit	Reserves (metric tons)	Est. Production Rate (metric tons/yr)*	Est. Duration of Production (yrs)	Necessary Startup Time (yrs)	Est. Price for Mine to be Economic ($/ton)**
Gasquet Mt.*** (Calif.)	222,546	12,367	20	7	367
Pilliken (Calif.)		12,827	---	---	368
Mouat/Bentow	2,229,758	48,473	46	3	231
Seaid Creek/ Emma Bell	10,512	1,168	9	3	763
SW Oregon Beach Sands	118,503	10,773	11	2	337
TOTAL	2,581,319				

*Tons of recoverable Cr.
**Estimated price based on total production costs per ton plus 20% return on investment.
*** Cr recovered as byproduct only.

Data compiled from Tables B-1, B-4, and 8-6. (Reserves = identified deposits from which minerals can be extracted with existing technology and under present economic conditions.)

U.S. Bureau of Mines, Mineral Commodities Summaries 1983, reports the year end price of Turkish high-chromium chromite at $110 per metric ton, Turkey. For South African high-iron chromite, the year end price was $48-52 per metric ton, South Africa.

Table 9. Principal United States PGM Deposits

Deposit	Reserves (million troy oz)*	Est. Production Rate (oz/yr)	Est. Duration of Production (yrs)	Necessary Startup Time (yrs)	Est. Price for Mine to be Economic ($/oz)
Goodnews Bay	1	50,000	20	Operating	400 (current)
Stillwater	10	500,000	20	2.5	450 (est.)
Duluth**	5	100,000	50	10+	400
TOTAL	16				

---

*Reserve data from USBM Mineral Commodity Profiles, 1983, Platinum Group Metals. USBM estimate of reserve base includes resources that are currently economic and marginally economic reserves and some that are currently subeconomic; therefore, figures may include PGM deposits usually categorized as resources rather than reserves.

** PGM production only as byproduct.

The U.S. Bureau of Mines, Mineral Commodities Summaries 1983, reports the 1982 average producer price of platinum was $475/ounce and palladium was at $110/ounce.

Table 10. Principal United States Cobalt Deposits

Deposit	Reserves Proven (millions of lbs)	 Potential	Est. Production Rate (million lbs/yr)	Est. Duration of Production (yrs)	Necessary Startup Time (yrs)	Est. Price for Mine to be Economic ($/lb)*
Blackbird	70[1]	250[4]	4.6	15+[1]	1.5 (no refinery)[2] 3-4 (refinery)[1]	11[3]
Madison	20	20	2	10-20	2	20 (0.16)
Missouri Additional	10-15		1-1.5	10	2	20 (0.16)
Gasquet Mt.	22[5]	10.6[5]	1.98[6]	18[6]	2[6]	20[7] (23.00)

1, USFS, 1982, Blackbird Mine EIS.
2, Scheuering, 1982, pers. com.
3, Fiorini, 1981.
4, 21 m.t. "geologic indicated reserves" (Fiorini, 1981; 0.6% Co.
5, Kaiser Engineers Mine Plan, 1982, p. 2-2.
6, USFS, in press, Gasquet Mine EIS.
7, Hosken, 1981, testimony before Senate Banking, Housing, and Urban Affairs Committee. Project also depends on prices for nickel and chromium.

Data from statements by representatives from Anshutz and Noranda Mining Companies, hearings before the Committee on Banking, Housing, and Urban Affairs, United States Senate, 97th Congress, October 26, 1981.

*Figures in parentheses from Table 8-19. For Missouri lead belt deposits, figures indicate incremental operating costs for Co recovery.

The U.S. Bureau of Mines, Minerals Commodities Summaries 1983, reported a 1982 producer price range for cobalt cathodes of $12.50-17.26 per pound.

for the significant deposits, and Chapter VI considers longer-term trends in mining and processing technologies. Chapters VIII and IX integrate the information from preceding chapters into possible production scenarios for the years 1995 and 2010. Although necessarily brief, the report provides the background necessary to assess both the conditions, and potential success, of a U.S. domestic supply of the commodities of interest.

Given the distribution of world resources of chromium, nickel, cobalt, manganese, and platinum, and the facilities currently on line to meet market demand, a number of critical questions arise: (1) What U.S. deposits, if any, could be in production by the year 1995? (2) At what price will the five commodities of interest stimulate production from U.S. resources? (3) For how long would these resources be available at anticipated prices? In other words, to assess the ability of the United States to support its strategic and critical metal demand, in the event of a prolonged and widespread interruption of supply from the traditional resource centers, one must evaluate the deposits that could be called into production.

The projections we have made for the years 1995 and 2010 are based in large part on the date base summarized in the following tables. Tables 1 through 5 summarize major world reserves and resources for the five critical materials; they also indicate ore grade and deposit type. Tables 6 through 10 indicate reserves figures for U.S. deposits of these minerals and also provide timing, production rate and price information.*

---

* In using tables 6-10 some additional explanation is helpful in interpreting the price information contained in the last column of these tables. Price information was derived from publicly available U.S. Bureau of Mines cost data for mines of the size and type of the deposits summarized. The prices shown do not represent a site specific analysis of the actual price at which these particular deposits would go into production. The prices are useful in comparing the estimated range of costs for typical mines of these types to current market prices to identify those deposits that might be mined at or near the current price and to identify those deposits that would only be economically competitive at substantially higher prices.

U.S. Metal Production in the Year 1995

It is clear that by 1995, relatively few U.S. strategic or critical resources of interest in this study could be established as active mining properties, almost irrespective of market price.

Chromite

For chromite, the Stillwater Complex of Montana and the northern California podiform/disseminated chromite deposits are the only potential primary U.S. economic supply that might be available in the year 1995. Only these deposits could supply a sufficient resource base to support mining for a reasonable period of time at prices considerably higher than current world prices, but still within a potential subsidy price framework.

Nickel

With respect to nickel, only three (primary) U.S. resources may be available by 1995. The most important in the United States, the Hanna Mine at Riddle, Oregon, is capable of producing 15,000 tons per year of nickel metal (Johnson, 1982). Other nickel laterites, especially the laterites of the northern California area, including Gasquet Mountain, are of significantly lower grade; with the exception of Gasquet Mountain, none of the others could be considered as potential minable resources by the year 1995. Gasquet Mountain is a remote possibility for mining, but given the status of world nickel supply and the expected price structure for the metal over the next decade, it is unlikely that the Gasquet Mountain deposit can be economically operated by 1995. A third potential nickel mining site in the United States is the nickeliferous Duluth Complex in northeastern Minnesota. These copper/nickel ores, associated with the stratiform, or layered, basic intrusion, were evaluated in the late 60's and early 70's, and are a possible high priced resource that could be available by 1995 in the unlikely event of a massive nickel supply cut off from overseas. However, they can be mined only if copper supply and price also are appropriate in the metal market structure.

Cobalt

With respect to cobalt, the only U.S. resources that can be possibly placed in production between now and the year 1995 are the sulfide vein (or strataform) Blackbird deposit of central Idaho and the deposits in the East Missouri Lead Belt (Madison Mine), in which cobalt occurs in low but recoverable concentrations associated with the sulfide ores. The Madison deposit cobalt is tied to pyrite and is readily amenable to milling and processing techniques. At almost any foreseeable cobalt price by the year 1995, these are the only two deposits that appear to be mineable in the United States. The nickeliferous laterite deposits of northern California and southern Oregon also contain very small quantities of cobalt. However, the cobalt is generally not recoverable from these laterite deposits and, except for the Gasquet Mountain proposal, cobalt recovery from the U.S. nickeliferous laterites has not been seriously considered because of the low grade and difficulty in recovery.

Platinum

Potential platinum mining in the United States by the year 1995 is restricted to essentially two future sites, the Stillwater Complex of Montana and the Duluth Complex of northeastern Minnesota. The expectation is that Goodnews Bay, Alaska, will continue to be mined until 1995 and perhaps beyond. Stillwater is still the potential mining site that is closest to production, for it is economically within the current platinum price structure and that expected by the year 1995. Mining in the Duluth Complex, however, is again a highly unlikely event within the 1995 time frame. Platinum concentrations are not high enough, within any foreseeable price structure, to warrant a mining enterprise aimed at platinum as the primary metal. Platinum in the Duluth Complex would clearly be a byproduct and, therefore, depend upon copper and nickel markets.

Manganese

Within the next 12 years manganese mining is highly unlikely in the United States except as a possible secondary material from some other primary base or precious metal deposit. Ore-grades are too low and processing costs too high for manganese mining to compete economically within any reasonably anticipated price structure to the year 1995. World supplies are enormous, diverse, cheap, and rich enough to make it extremely unlikely that any major supply shortage could result even from the cut off of a few U.S. import sources. The mining of manganese in the United States, therefore, cannot supply even a small amount of U.S. annual requirements to the year 1995.

Nothing said above, in the 1995 scenario is likely to change based upon mining or processing cost reductions now in the development stream. Significant technological and cost-reduction breakthroughs aimed at the five metals of interest would, at best, be aimed at the 21st century. However, even these, should they become a reality, would be difficult to capture for the exclusive cost benefit of the U.S. deposits. Incremental improvement in mining and processing technology will surely occur; all deposits and smelters (recovery plants) will benefit, whether it be from increased metal recovery rates or reduced energy consumption. However, the benefits are not thought to improve the relative competitiveness of the U.S. deposits on the U.S. or world markets.

U.S. Metals Production in the Year 2010

Strategic mineral development in the United States in the year 2010, especially for the metals considered in this study, provides a different set of problems than those considered for 1995. First, a scenario for 1995 nearly limits analysis to those deposits that are already discovered or known mineralized areas; however, a 2010 scenario must also consider potential new discoveries in new areas containing the minerals of interest. The major world-class deposits of the metals of interest at the highest grade have already been identified; most are producing ores today and will continue to supply the major part of the mineral demand in 2010. (See Chapters II and III for a brief discussion of the size and grade of the resource base.) Our estimates indicate that platinum will probably be in production from Alaska and the Stillwater Complex by 1995, and we thus assume that these areas will still be in production in 2010. It is possible that presently known U.S. cobalt deposits may also come on line by 2010, depending upon metal price, political access, and environmental variables; these possibilities are discussed in the scenarios developed in Chapter IX. It is unlikely that chromite, nickel and manganese will be produced in 2010 without significant government intervention in the market.

Second, the projection of economic, political, and material conditions in the world to the year 2010 is exponentially more difficult to forecast than for 1995. Although one might project some degree of world stability for the near-term, to project 25 to 30 years in advance is to supply the reader with a number of assumptions, any of which can be credibly challenged. It is clear, however, that the potential for developing most U.S. deposits is contingent upon interruption of foreign supplies or other political considerations. Such considerations will determine whether substantial future mining of the metals of interest will occur even at elevated costs and prices.

Third, although our analysis suggests that current market share of production cannot be significantly changed within the next 30 years, that possibility cannot be absolutely refuted. In addition, a lead time of 25 to 30 years also provides the opportunity to assess a substitution strategy that may make traditional uses of one or more of the studied metals obsolete. We do not intend to provide an analysis of recycling or substitution, but rather we assume that the metals of interest will be required in essentially traditional ways in the world economy. If that is the case, there are known world resources and reserves that can provide the bulk of world demand, resources that are already identified and some of which are perhaps only marginally uneconomic today.

We assume increased demand for the commodities of interest to the year 2010, ignoring the potential for substitution and conservation due simply to an expanding world population and, increased economic activity yielding a modest increase, perhaps 1% to 2% per year in growth rate. The combintion of the per capita increase in consumption and increasing world population suggest that in the year 2010 more of the commodities of current interest will be required. There is some reason to believe, however, that we may see significant changes in the ways in which these metals are used within the industrial community. Some of those shifts may temporarily reduce the demand of the materials over a short period of time; however, over the next 25 years it seems more likely that world demand for these commodities will be at levels 50% to 75% higher than those we see today as more of the nonindustrial world undergoes economic transition. (Despite the increase, presently known deposits outside the United States can easily meet this need under assumed normal economic conditions.)

Fourth, we have shown in Chapters VI and VII that it is very unlikely that significant mining or metallurgical breakthroughs in the mining and processing of the low-grade, high-cost, or low-quality ores that generally exist in the United States will affect the expected supply picture. Most improvements that can be introduced in mining and metallurgy by 2010 cannot be exclusively applied to the U.S. resource base and therefore U.S. deposits are unlikely to become substantially more economic vis a vis other world ores.

In the year 2010 we consider two scenarios of interest for each of the commodities discussed. One is that the world mineral flow will essentially proceed under a scenario of business-as-usual in which the lowest cost resources will be exploited first: under this scenario there will be free international trade at least within the western industrialized societies and the investment in supplies between now and the year 2010 will be conditioned essentially on least-cost decisions and open-mineral markets without cartels. The second scenario to be considered will be one in which overseas supplies of commodities of interest are cut off for an indefinite period of time and the United States is forced to rely on domestic supply in order to sustain its material enterprises. This second scenario will consider only the possibilities of relying on our domestic supply as well as Canadian resources and will not try to defend the proposition of a cut-off as being either likely or unlikely in terms of world conditions.

In summary, then, the strategic and critical minerals picture for the United States appears as follows for the year 2010. Under a scenario of business-as-usual, it is unlikely that American deposits will produce any primary ore chromite, manganese, or nickel; only small amounts of platinum and some cobalt could be produced, however. Few domestic deposits will be able to compete economically with materials available on the world market. It is only in a scenario of import restriction, or cut-off, that the United States could perhaps develop domestic resources for the first three metals, or expand production of platinum and, to a lesser extent, cobalt.

Chromium

In a nonimport scenario for the year 2010, we may be able to produce low-grade chromite ores from the Stillwater Complex of Montana and the podiform/disseminated deposits of northern California and southern Alaska. We do not anticipate production levels that could meet normal U.S. demand, and significant reductions of chromite and ferrochrome use would have to occur to balance supply and demand from domestic resources. In addition, that supply would not come on-line without a doubling or 2.5-fold increase in the price, based on 1982 dollars.

Cobalt

For cobalt, ores could be available in 2010 at the equivalent 1982 market price of $20.00 a pound from Blackbird, Idaho, and at $25.00 pound from the East Missouri Lead Belt, where both cobalt and nickel would be recovered from the tailing sulfides mined with the primary lead-zinc ores. Some small amount of cobalt is recoverable from the nickeliferous laterites in northern California and would owe its market availability to the economics of nickel rather than cobalt. Small amounts of cobalt could also come as a byproduct from copper/nickel mining in the Duluth Complex. In the year 2010, U.S. supplies of cobalt would not be able to provide the total domestic demand because the resource base outlined above is probably not capable of meeting the required level of production.

Nickel

Again, in the year 2010, nickel ores below a price level of about $6.00 a pound, in 1982 dollars, would not be produced in the United States. However, at that price, nickeliferous laterites from northern California, the nickel ores of the Duluth Complex, and the nickel byproduct metal of the east Missouri Lead Belt might be available for production. A price of nickel, approximately twice the current price in real terms, could bring a significant amount of nickel to the U.S. market from domestic resources. A total cut-off of foreign supply might add the nickeliferous copper ores of brady Glacier and other S.E. Alaska deposits to the U.S. supply base, continue the expansion of the Duluth Complex, and bring on the production of the lower-grade nickeliferous laterites of northern California. Essentially, the geologic sources would remain the same, but expanded mining and a larger supply would be available at prices somewhat greater than the $6.00 threshold price.

Manganese

Manganese ores could be available at 5 to 10 times the current price of manganese, primarily from Chamberlain, South Dakota, the deposits of Aroostook County and Maple/Hovey Mountain, Maine, and the Cuyuna Range of Minnesota. These latter two ores might pose some beneficiation and metallurgical application problems, which could easily be resolved in the nearly 30 years to the 2010 benchmark. Under a business-as-usual scenario, manganese will not be mined in the United States in the year 2010.

Platinum

Platinum ores will be provided by the Stillwater Complex and rich placer deposits of Alaska, from now until the year 2010. The ores are clearly available as part of the U.S. resource base and can provide a small amount, perhaps up to 5% to 10%, of U.S. domestic demand in the year 2010. A critical cut-off of platinum supply on a worldwide basis would lead to the development of platinum from the Duluth Complex and the expansion of placer mining in Alaska. Platinum in the Duluth Complex would only be recovered along with nickel and copper and therefore, the primary ore metals would control the availability of platinum.

It is conceivable that total domestic platinum needs could be supplied (for a short period of time) by the mining of the Duluth Complex, along with the known placer resources in Alaska and the Stillwater ores. It would require, however, that the Duluth Complex produce a great mining enterprise, one as yet unequaled in the United States. The constraints to an operation of that size are many, especially in terms of environmental and institutional limits and aternate use patterns of the areas of interest. It is doubtful, therefore, that the United States will ever be self-sufficient in platinum, regardless of expected trends in PGM prices so long as large volumes are needed for catalytic converters and large industrial uses.

Chapter 1

BIBLIOGRAPHY

Johnson, M., 1982, Personal communication (Management Engineer, Hanna Mining Co., Riddle, Ore.)

U.S. Bureau of Mines and U.S. Geological Survey, 1976, Principles of the Mineral Resource Classification System of the U.S. Bureau of Mines and U.S. Geological Survey, Geological Survey Bulletin 1450-A. Washington, D.C.: U.S. Government Printing Office.

Table 1. Nickel Reserves and Resources (000 short tons)

	RESERVES			RESOURCES		
Country	Ni Reserves*	Ore Grade (%)	Deposit Type	Ni Resources**	Ore Grade*** %	Deposit Type
United States	---	---	---	14,900	1.5-0.3	Laterite; nickel sulfide
New Caledonia	15,000	1-3	Laterite	31,000	1.8	Laterite
Philippines	5,700	0.4-2.2	Laterite	10,600	0.8	Laterite
Cuba	3,400	1.4	Laterite	14,200	1.0	Laterite
Canada	8,700	1.5-3	Nickel sulfide	12,500	1.0	Nickel sulfide
USSR	8,100	1.5	Laterite Nickel sulfide	13,200	0.6-1.5	Laterite; nickel sulfide
Indonesia	7,800	1.8	Laterite	55,000	0.9	Laterite
Australia	5,600	1.5-3	Laterite Nickel sulfide	3,200	1.2-1.5	Laterite; nickel sulfide
Zimbabwe	661 (UN)	---	---			
Greece	2,400 (UN)	---	---			
TOTAL****	57,361			154,600		

---

*Reserve figures and ore grade from USBM Mineral Commodity Summaries, 1983. USBM estimate of reserve base includes resources that are currently economic, marginally economic reserves and some that are currently subeconomic; therefore, figures may include nickel deposits usually categorized as resources rather than reserves.

**Mineral Facts and Problems, USBM, 1980. (Resources = specifically identified deposits whose contained minerals cannot be recovered with existing technology and economic conditions.)

***Professional Paper 820, USGS, 1973.

****Total represents major reserves and resources; other small miscellaneous deposits are not included.

Table 2. Cobalt Reserves and Resources (metric tons)

Country	Co Reserves*	RESERVES Ore Grade (%)**	Deposit Type**	Co Resource*	RESOURCES Ore Grade (%)	Deposit Type**
Philippines	181,000 (425,000)	.03-0.12	Laterite	23,000 (1,750,000)	.03-0.12	Laterite
Cuba	181,000 (800,000)	.03-0.12	Laterite	953,000 (300,000)	.03-0.12	Laterite
New Caledonia	91,000 (385,000)	.03-.05	Laterite	295,000 (450,000)	.03-0.05	Laterite
Indonesia	--- (565,000)		Laterite	--- (300,000)		Laterite
Zaire	1,179,000 (450,000)	0.1-0.45	Stratabound copper	227,000 (230,000)	0.1-0.45	Stratabound copper
Zambia	363,000 (300,000)	.09-0.4	Stratabound copper	227,000 (50,000)	.09-0.4	Stratabound copper
USSR	227,000 (200,000)	.03-0.1	Laterite Magmatic sulfide	23,000 (150,000)	.03-0.1	Laterite Magmatic sulfide
Australia	45,000 (135,000)	.08-0.5	Laterite Magmatic sulfide	249,000 (165,000)	.08-0.5	Laterite Magmatic sulfide
Canada	27,000 (30,000)	.03-0.11	Magmatic sulfide	227,000 (220,000)	.03-0.11	Magmatic sulfide
United States	---	---	---	770,000	.01-0.6	Laterite; magmatic sulfide
TOTAL***	2,294,000			2,994,000		

---

*Data from USBM, 1980, Mineral Facts and Problems. Figures in parentheses from Berndesanstalt fur Geowissenchaften und Rohstoffe (Hanover), 1978. (Definitions: Reserves = identified deposits from which minerals can be extracted with existing technology and under present economic conditions; conditional resources = specifically identified deposits whose contained minerals cannot be recovered with existing technology and economic conditions.

**Ore grade and deposit type apply to combined reserves and resources (Foose, 1981). ***Major reserves and resources:very small deposits not included.

Table 3. Chromium Reserves and Resources (000 long tons)*

Country	RESERVES Cr Reserves	 Ore Grade (%)	 Deposit Type	RESOURCES Cr Resources	 Ore Grade (%)	 Deposit Type
United States	---	---	---	350	46	Podshaped
Brazil	6,000	40-46	Stratiform	5,150	40-46	Stratiform
Cuba	250	46	Podshaped	1,100	46	Podshaped
India	7,000	40-46	Stratiform Podshaped	6,000	40-46	Stratiform Podshaped
USSR	21,000(?)	40-46	Podshaped Stratiform	22,000	40-46	Stratiform Podshaped
Finland	10,000	40-46	Stratiform	5,000	40-46	Stratiform
Turkey	5,000	46	Podshaped	5,000	46	Podshaped
Malagasy Republic	5,000	46	Stratiform(?)	6,000	40-46	Stratiform(?)
Zimbabwe	550,000	40-46	Stratiform Placer	500,000	40-46	Stratiform Placer
South Africa	1,050,000	40-46	Stratiform	2,050,000	40-46	Stratiform
Philippines	4,700	46	Podshaped	2,500	46	Podshaped
Iran	1,500	46	Podshaped	1,000	46	Podshaped
TOTAL**	1,660,450			2,604,100		

*Data from USGS Professional Paper 820, U.S. Mineral Resources, p. 117. (Reserves = identified deposits from which minerals can be extracted with existing technology and under present economic conditions; conditional resources = specifically identified deposits whose contained minerals cannot be recovered with existing technology and economic conditions.

**Total represents major reserves and resources; other small miscellaneous deposits not included.

15E

Table 4. Manganese Reserves and Resources (000 metric tons)*

	RESERVES			RESOURCES		
Country	Mn Reserves	Ore Grade (%)	Deposit Type	Mn Resources	Ore Grade (%)	Deposit Type
South Africa	901,600	29-39	Mixed carb/oxide, oxide, carb (sed, surficial/resid)	1,960,000	27-36	Carb/oxides/silicate (sed, surf/resid)
USSR	2,278,800	18.5-25	Oxide, carb, mixed carb (sed, surficial/resid)	152,000	4-25	Mostly carb/silicate (sed/meta/resid)
United States	---	---	---	925,000	4-9	Carb/silicate, silicate/carb (sed/meta); oxides (sed?/hypogene); oxide/silicate/carb (sed/resid); oxide/carb (sed/meta/resid)
Gabon	194,000	50	Oxide (sed/surf-resid)	---	----	Carb/silicate (sed/meta)
Canada	---	---	---	193,000	9	Carb/silicate (sed/meta)
Australia	213,000	48	Oxide (sed)	100	52	Oxide (sed)
Brazil	60,609	43 (inferred only)	Oxide (lump), oxide (pellets); Carbonate (sed/meta,surf/resid)	69,200	20-47	Oxide (sed/resid)
India	53,650	38-48	Oxide, silicate (sed) Oxide (sed/meta)	---	----	----
Hungary	---	---	---	215,000	20.5	Carb (sed/exhal)
TOTAL	3,701,659			3,514,300		

---

*Data from NAS National Materials Advisory Board, 1981, Manganese Reserves and Resources of the World and their Industrial Implications. (Reserves = identified deposits from which minerals can be extracted with existing technology and under present economic conditions; resources = specifically identified deposits whose contained minerals cannot be recovered with existing technology and economic conditions.

**Total represents major reserves and resources; other small miscellaneous deposits not included.

Table 5. PGM Reserves and Resources

	RESERVES			RESOURCES		
Country	PGM Reserves (mil.tr. oz)*	Ore Grade (oz/ton)**	Deposit Type**	PGM Resources*** (mil.tr.oz)	Ore Grade	Deposit Type
South Africa	970	.09-.35	Stratiform	1,430	NA	Stratiform
USSR	200	.305	Stratiform, alpine, placer, sulfide	200	NA	Stratiform, alpine, sulfide, placer
Canada	9	.026-.11	Stratiform	7	NA	Stratiform, sulfide
Colombia	< 1	---	Placer	4	NA	Placer
United States	16	.02-.45	Stratiform, placer	285	NA	Stratiform, placer, sulfide
Zimbabwe	--	---	---	100	NA	Stratiform
TOTAL	1,195			2,026		

---

*Data from USBM Mineral Commodity Profiles, 1983, Platinum-Group Metals. Estimate of reserve base includes resources that are currently economic, marginally economic reserves, and some that are currently subeconomic; therefore, figures may include nickel deposits usually categorized as resources rather than reserves.

**Data from USGS Professional Paper 820, 1973.

***Data from Mineral Facts and Problems, USBM, 1980. Resources = specifically identified deposits whose contained minerals cannot be recovered with existing technology and economic conditions.

Table 6. Principal United States Nickel Deposits

Deposit	Reserves (metric tons)	Est. Production Rate (metric tons/yr)	Est. Duration of production (yrs)	Necessary Startup Time (yrs)	Est. Price for Mine to be Economic ($/lb)**
Nickel Mt. (Oregon)	100,000	5,000	20	0	3.02-4.55
Gasquet Mt. (Calif.)	157,909	8,773	21	7	3.02-4.55
Madison (Mo.)	7,000	700	10	3	12.51
Brady Glacier (Alaska)	128,953	11,723	11	9	4.89
Minnamax (Minn.)	480,643	11,723	41	3	4.89
TOTAL					

---

*Tons of recoverable Ni.

**Estimated price based on total production costs plus 20% return on investment.

Data compiled from Tables 8-7, 8-10, 8-11, 8-12, and 8-13. (Reserves = identified deposits from which minerals can be extracted with existing technology and under present economic conditions.)

The 1982 average producer price for nickel according to U.S. Bureau of Mines, Mineral Commodities Summaries 1983, was $3.20/lb.

Table 7. Principal United States Manganese Deposits

Deposit	Reserves (metric tons)	Est. Production Rate (metric tons/yr)*	Est. Duration of Production (yrs)	Necessary Startup Time (yrs)	Est. Price for Mine to be Economic ($/long ton unt unit)*
Maggie Mine (Ariz.)	300,000	30,000	10	4	10.44
Sunnyside (Colo.)	600,000	30,000	20	4	10.44
Maple/Hovey (Maine)	8,540,000	140,000	61	6	6.00
Cuyuna North Range (Minn.)	1,960,000	140,000	14	4	6.00
TOTAL	11,400,000				

*Tons of recoverable Mn.
**Estimated price based on total production costs per long ton unit plus 20% return on investment.

Data compiled from Tables 8-20 and 8-23. (Reserves = identified deposits from which minerals can be extracted with existing technology and under present economic conditions).

U.S. Bureau of Mines, Minerals Commodities Summaries 1983, reports a 1982 average import price range for 46-48% manganese ore of $1.58-1.68 per lon ton unit cont. mn. c.i.f., U.S. ports.

Table 8. Principal United States Chromium Deposits

Deposit	Reserves (metric tons)	Est. Production Rate (metric tons/yr)*	Est. Duration of Production (yrs)	Necessary Startup Time (yrs)	Est. Price for Mine to be Economic ($/ton)**
Gasquet Mt.*** (Calif.)	222,546	12,367	20	7	367
Pilliken (Calif.)		12,827	---	---	368
Mouat/Benbow	2,229,758	48,473	46	3	231
Seaid Creek/ Emma Bell	10,512	1,168	9	3	763
SW Oregon Beach Sands	118,503	10,773	11	2	337
TOTAL	2,581,319				

*Tons of recoverable Cr.

**Estimated price based on total production costs per ton plus 20% return on investment.

*** Cr recovered as byproduct only.

Data compiled from Tables 8-1, 8-4, and 8-6. (Reserves = identified deposits from which minerals can be extracted with existing technology and under present economic conditions.)

U.S. Bureau of Mines, Mineral Commodities Summaries 1983, reports the year end price of Turkish high-chromium chromite at $110 per metric ton, Turkey. For South African high-iron chromite, the year end price was $48-52 per metric ton, South Africa.

Table 9. Principal United States PGM Deposits

Deposit	Reserves (million troy oz)*	Est. Production Rate (oz/yr)	Est. Duration of Production (yrs)	Necessary Startup Time (yrs)	Est. Price for Mine to be Economic ($/oz)
Goodnews Bay	1	50,000	20	Operating	400 (current)
Stillwater	10	500,000	20	2.5	450 (est.)
Duluth**	5	100,000	50	10+	400
TOTAL	16				

---

*Reserve data from USBM Mineral Commodity Profiles, 1983, Platinum Group Metals. USBM estimate of reserve base includes resources that are currently economic and marginally economic reserves and some that are currently subeconomic; therefore, figures may include PGM deposits usually categorized as resources rather than reserves.

** PGM production only as byproduct.

The U.S. Bureau of Mines, Mineral Commodities Summaries 1983, reports the 1982 average producer price of platinum was $475/ounce and palladium was at $110/ounce.

15K

Table 10. Principal United States Cobalt Deposits

Deposit	Reserves Proven (millions of lbs)	Reserves Potential (millions of lbs)	Est. Production Rate (million lbs/yr)	Est. Duration of Production (yrs)	Necessary Startup Time (yrs)	Est. Price for Mine to be Economic ($/lb)*
Blackbird	70[2]	250[4]	4.6	15+[2]	1.5 (no refinery)[2] 3-4 (refinery)[2]	11[3]
Madison	20	20	2	10-20	2	20 (0.16)
Missouri Additional	10-15		1-1.5	10	2	20 (0.16)
Gasquet Mt.	22[5]	10.6[5]	1.98[6]	18[6]	2[6]	20[7] (20.00)

1, USFS, 1982, Blackbird Mine EIS.
2, Scheuering, 1982, pers. com.
3, Fiorini, 1981.
4, 21 m.t. "geologic indicated reserves" (Fiorini, 1981; 0.6% Co.
5, Kaiser Engineers Mine Plan, 1982, p. 2-2.
6, USFS, in press, Gasquet Mine EIS.
7, Hosken, 1981, testimony before Senate Banking, Housing, and Urban Affairs Committee. Project also depends on prices for nickel and chromium.

Data from statements by representatives from Anshutz and Noranda Mining Companies, hearings before the Committee on Banking, Housing, and Urban Affairs, United States Senate, 97th Congress, October 26, 1981.

*Figures in parentheses from Table 8-19. For Missouri lead belt deposits, figures indicate incremental operating costs for Co recovery.

The U.S. Bureau of Mines, Minerals Commodities Summaries 1983, reported a 1982 producer price range for cobalt cathodes of $12.50-17.26 per pound.

Chapter II

DESCRIPTION AND USE OF METALS OF INTEREST

COBALT

Description and Uses

Cobalt is a silver-gray metal which is resistant to both heat and fatigue over long periods of time. It has an atomic number of 27, an atomic weight of 58.94, a specific gravity of 8.9 and a melting point of 1495°C. The majority of world-produced cobalt is used in the United States. Almost 80% goes into metallurgic applications, primarily in high temperature, high strength, magnetic and super-alloy materials. Cobalt is also used in the production of various steels, and is alloyed with other metals in small quantities.

Types of Cobalt-Bearing Deposits

A number of authors (Brobst and Pratt, 1973; Foose, 1982; USBM, RI 8848) have characterized the following important characteristics of U.S. cobalt resources. First, cobalt occurs in primarily low grade concentrations in a variety of geologic environments, most of which are not exploitable today, or in the foreseeable future. Secondly, cobalt is a byproduct metal in some important geologic environments; therefore, its availability is tied to mining other materials. Thirdly, except in those few deposits where cobalt is the primary metal of interest, the supply will be relatively unaffected by small to modest changes in cobalt price because other metal prices will dictate the economies of the deposit; cobalt availability will be dictated primarily by the price of the associated primary metals. There are exceptions to this assessment: in particular, the potential for mining relatively high grade cobalt ores in central Idaho (Blackbird district) and recovering low grade cobalt from cobaltiferous pyrites of the east Missouri Lead Belt.

Worldwide, five or six different geologic environments are known to contain concentrations of cobalt higher than average crustal concentrations and are potentially mineable. These deposit types, on a worldwide basis, are discussed below in order of importance with respect to the current mining.

Stratabound Copper/Cobalt Deposits

The most notable stratabound copper/cobalt deposits are the ores of Zaire and Zambia, which produce the majority of the world's cobalt. The United States receives 60% to 70% of its cobalt supply from these deposits. The ores are associated with fine-grain sedimentary rocks (shales and carbonates) of Proterozoic age and are the richest cobalt ores known. They reflect an environment of deposition widespread in rocks of that age, many of which contain copper ores like those of Zaire and Zambia. But in no other place is the cobalt content reported to be as high or as continuous as those of central Africa. Stratabound deposits containing cobalt resources account for between 55% to 60% of world cobalt production; however, these ore types contain only about 15% to 20% of the cobalt resource base in known deposits (Table 2-1).

The metal content of the African ore-bearing strata varies considerably, ranging from 0.1% to 0.5% cobalt to as much as 1% to 2% .pacobalt. The Zambian ore grades have a more uniform cobalt concentration and for the most part average 0.5% cobalt. Copper grades in both Zaire and Zambia range from between 4% and 6%.

These stratabound copper/cobalt deposits of Zaire and Zambia are the best known and best studied in the world. There well may be other places in the world where such deposit types might be discovered. Unfortunately, even though rocks of similar age and character, and containing copper and other metals in

TABLE 2-1

WORLD COBALT PRODUCTION AND RESERVES BY GEOLOGIC TYPE
(in percent)

		Reserves	
Geologic Type	Production	Proven and Probable	Potential
Hypogene Deposits Associated with Mafic Intrusive Igneous Rocks (Sudbury Type)	21	14	14
Stratabound Deposits (Zambian - Zairian Type)	55	21	7
Hydrothermal Deposits (Blackbird Type)	7	1	1
Laterite Deposits (Gasquet Mountain Type)	17	64	78
TOTAL	100	100	100

Source: Dames & Moore, 1980, p. A-16. Data adapted from
Kruszona, et al., 1979, p. 52

associations not unlike the African deposits, are known to occur elsewhere in the world, nowhere are rocks found to be enriched in cobalt. Exploration for world class stratabound deposits continues, but in those places where a significant effort has been recently made, such as Belt Series rocks in Idaho, Montana, and Washington and in similar age rocks of Georgia, cobalt is yet to be found in potentially important economic concentrations.

Laterite Deposits

Cobaltiferous, nickel laterite deposits occur in many areas throughout the world and the source rocks are often exposed by faulting near the edges of active (subducting) crustal plates. These deposits usually contain up to 50% iron, 1% to 2% nickel and from 0.01% to 0.1% cobalt. They form primarily by chemical weathering of ultramafic rocks, usually periodotites and serpentinites. Significant cobalt bearing nickel laterites are known to occur in Australia, Cuba, Indonesia, New Caladonia and the Philippines. Others of potential economic value occur in northern California and southern Oregon, but are of much lower grade than their worldwide counterparts.

Circulating ground water usually distributes the cobalt in a laterite in the upper iron-rich parts of the weathering profile, where it is commonly found associated with manganese. However, inasmuch as manganese distribution in the upper weathering profile of the laterites is usually discontinuous, cobalt concentrations, particularly when compared to nickel and the nickel/iron ratio, can be highly erratic within these laterite deposits.

Cobalt occurs as a minor or accessory constituent in most of the world nickel laterites (65% to 75%); however, there is no reason to believe that regardless of mining or market conditions cobalt will be recovered from all of them (Tables 2-1 and 2-2). As a matter of fact the usual case is not to recover the cobalt from laterites that are commercially nickel bearing. The potential of future production of cobalt from laterites is, therefore primarily, but not absolutely, tied to the level of future nickel production. Given their low grade, increasing cost of production, and other factors associated with cobalt recovery from nickel laterites, this source only accounted for 15% to 20% of world cobalt production in the 1970's (see Table 2-1).

Magmatic Sulfide

During the crystalization of ultramafic and mafic magmas (molten rock) in the presence of sulfur, droplets of metal sulfides separate from the crystalizing magma, usually accumulate near the bottom of the magma chamber and can form sulfide ores of iron, nickel, copper, cobalt and platinum group metals. Such concentrations can be economically important and their significant mineralogy occurs as iron and copper sulfides (Table 2-3). Cobalt concentrations in these ore environments are usually low, and range from 0.01% to 0.5%, however, grades as high as 5% cobalt are known to exist in small bodies. The most important known deposits of this type worldwide are the nickel/copper, cobalt bearing, ores of the Sudbury Complex in Ontario, the Abitibi and Thompson districts also of Canada, the Bushveld Complex of South Africa, and the Noril'sk intrusive of the USSR. Within the United States, magmatic copper/nickel ores, with very minor cobalt, are found in the Duluth Complex of Minnesota, the Stillwater Complex of Montana, and the Brady Glacier and other deposits S.E. Alaska.

In the best examples of this ore type worldwide, cobaltiferous copper/nickel sulfide deposits average 0.07% cobalt and approximately 1.5% of

Table 2-2
Resources Estimate

Country	Resources cobalt metal (1000 tons)	Grade of deposit	Type of deposits
Phillippines	2,000	.03-.12	laterite
Cuba	1,100	.03-.12	"
New Caledonia	950	.03-.05	"
Indonesia	850	.03-.15	"
Zaire	700	0.1-.45	stratabound copper
Zambia	350	.09-.4	"
U.S.S.R.	350	.03-.1	laterite and magmatic
Australia	300	.08-.5	"
Canada	250	.03-.11	magmatic
Others	1,200		
TOTAL	7,750		

After Foose, 1982.

TABLE 2-3

MAJOR COBALT MINERALS OF ECONOMIC IMPORTANCE

Mineral	Formula	Percent Cobalt
Linnaeite	Co_3S_4	58.0 (theoretical)
Siegenite	$(Co,Ni)_3S_4$	20.4 - 26.0
Carrollite	$(Co_2Cu)S_4$	35.2 - 36.0
Cobaltite	$(Co,Fe)AsS$	26.0 - 32.4
Safflorite	$(Co,Fe)As_2$	13.0 - 18.6
Glaucodot	$(Co,Fe)AsS$	12.0 - 31.6
Skutterudite	$(Co,Fe)As_3$	10.9 - 20.9
Heterogenite	$CoO(OH)$	64.1 (theoretical)
"Asbolite"	(Manganese oxides + Co)	0.5 - 5.0
Erythrite	$(Co,Ni)_3(AsO_4)_2 \cdot 8H_2O$	18.7 - 26.3
Gersdorffite	$(Ni,Co)AsS$	(low)
Pyrrhotite	$(Fe,Ni,Co)_{x-1Sx}$	1.00 (maximum)
Pentlandite	$(Fe,Ni,Co)_9S_8$	1.50 (maximum)
Pyrite	$(Fe,Ni,Co)S_2$	13.00 (maximum)
Sphalerite	$Zn(Co)S$	0.30 (maximum)
Arsenopyrite	$Fe(Co)AsS$	0.38 (maximum)
Manganese oxide minerals	Variable	0.10 - 1.00 (or more

Source: USGS Professional Paper 820, 1973, p. 146.

both copper and nickel. Mineralization extends to depths of over 10,000 feet below the surface and is largely restricted to the margins of an intrusive complex as pods, disseminations and vein bodies in the layered lower norites. These ores originate from the partial melting of the mantle deriving a large mafic magma source, or alternatively, by the interactions of crustal and mantle melting generated by the collision of a large meteorite fragment with a continental land mass. In the latter explanation, such collisions are infrequent through geologic time, and the major stratiform layered intrusives of great size which contain these ores are also rare.

A variant of the magmatic sulfide deposits are contact metamorphic ores that are associated with mafic intrusive rocks. The best known examples of this in the United States are the cobalt-bearing sulfides associated with diabase sills in Lebanon, Pennsylvania. At the Cornwall deposit, diabase intrudes limestone forming a contact deposit of iron ore on the boundary between the diabase and the limestone. Here, pyrite containing up to 1.4% cobalt occurs in amounts that have been commercially recovered in the past. This ore type is not abundant worldwide and relatively few if any deposits are currently in production for cobalt. Another potential cobalt producing environment involves massive sulfide deposits in metamorphic or metasedimentary rocks.

The host rocks are generally of volcanic/sedimentary origin and the ores are principally rich in zinc, copper, and iron, and the cobalt occurs in pyrite in low concentrations, usually less than 0.5% of the pyrite. If pyrite is 10% of the ore, then the cobalt ore grade for the whole deposit is only 0.05%.

Hydrothermal Deposits

Important, but not abundant, cobalt resources are derived when heated aqueous fluids containing cobalt and other metals circulate through igneous or sedimentary rocks. These ores concentrate along fractures, producing vein deposits of cobalt sulfides and arsenides with associated values of copper, lead, zinc, silver, and gold. The silver-cobalt arsenide deposits of Ontario, Canada, and the lead-zinc-silver deposits of the Bawdwin district of Burma are examples of these ore occurrences. Domestically, hydrothermal solutions that produced potentially economic cobalt-bearing minerals are best known in the southeast Missouri Lead Belt. Here, the cobalt appears to be closely associated with pyrite in the ores and, although these minerals are present in low concentrations, they have the potential to be economically recovered.

The richest of the U.S. cobalt deposits are those in the Blackbird area of central Idaho. The Blackbird cobalt-bearing sulfide deposit appears localized along linear belts that are perhaps as intimately related to the stratigraphy of the host rocks as they are to cross cutting faults or shear zones. The Blackbird ores may, in fact, be a variation of stratiform sulfide deposits. They contain the highest potential ore grades in the U.S., averaging 1.3% copper and 0.55% cobalt. The mineralization at Blackbird is primarily cobaltite together with chalcopyrite and pyrite. Mineralization is mainly contained within one rock unit over a known distance of at least 2.2 km. Other hydrothermal veins and associated replacement deposits mined mainly for their cobalt content include the Bou Azzer district veins of Morocco.

NICKEL

Description and Uses

Nickel is a soft, silvery-white metal that has a high degree of malleability and ductility and at the same time is highly prized as an anti-corrosive substance. It has an atomic number of 28, an atomic weight of 58.71, a melting point of 1453°C, and a density of 8.9. Nickel is highly resistant to corrosion by both acids and alkalies. In the highly pure form, most nickel is utilized as pellets, cathodes and bound in nickel alloys. In its powder form, nickel is also used in the chemical industry; nickel briquets are used for the fabrication of nickel oxide batteries. Nickel in the form of ferro-nickel and oxide salts is mainly used to form alloys in stainless steel.

Types of Nickel-Bearing Deposits

Nickel is one of the more abundant elements in the earth's crust, and it occurs enriched in a variety of geologic environments. Significant concentrations of nickel, however, are localized in ultramafic to mafic igneous rocks and in the residual weathering profile of these materials. The major primary mineral compositions associated with nickel ores are nickel sulfides and arsenides, as well as nickeliferous bearing iron sulfide minerals such as pyrite and pyrrhotite (Table 2-4).

TABLE 2-4

PRINCIPAL NICKEL MINERALS

Mineral	Formula	Occurrence
Native nickel-iron	Ni_3Fe	Rare
Pentlandite	$(Fe,Ni_9S_8$	Common
Bravoite	$(Fe,Ni)S_2$	Minor; secondary
Violarite	Ni_3FeS_4	Minor
Vaesite	NiS_2	Rare
Polydymite	Ni_3S_4	Rare
Millerite	NiS	Minor; secondary
Heazlewoodite	Ni_3S_2	Rare
Siegenite	$(Ni,Co)_3S_4$	Minor
Linnaeite	$(Co,Fe,Ni)_3S_4$	Minor
Geradorffite	$NiAsS$	Minor
Niccolite	$NiAs$	Minor
Rammelsbergite	$NiAs_2$	Rare
Chloanthite	$(Ni,Co)A_{3-x}$	Rare
Smaltite	$(Co,Ni)A_{3-x}$	Rare
Skutterudite	$(Co,Ni)A_3$	Rare
Maucherite	$Ni_{11}As_8$	Rare
Breithauptite	$NiSb$	Rare
Ullmannite	$NiSbS$	Rare
Parkerite	$Ni_3Bi_2S_2$	Rare
Annabergite	$Ni_3(AsO_4)_2 \cdot 8H_2O$	Rare; secondary
Morenosite	$NiSO_4 \cdot 7H_2O$	Rare; secondary
Zaratite	$NiCo_3 \cdot 2Ni(OH)_2 \cdot 4H_2O$	Rare; secondary
Garnierite	Nickel-magnesium hydrosilicate	Common; secondary.

Source: USGS Professional Paper 820, 1973, p. 438.

Straciform/Massive Sulfides

Two major types of nickeliferous ores occur. The first are the sulfide deposits. These ores are commonly found associated with basic or ultrabasic intrusives, and the major ore bodies are usually disseminated layers of sulfides or vein bodies which cut the basic or ultrabasic host rocks and the surrounding country rock. Nickel sulfide deposits, therefore, are usually either elongate and tabular, following either the sheet-like or straciform nature of the intrusive itself, or vein-like, extending through or out from the intrusive body.

Through the process of liquid immiscibility, nickel sulfide liquids form in the magma chamber, segregate and fall to the margins and the bottom of the intrusive, cool and crystallize later than the surrounding country rock materials. At times the sulfide liquid can be injected along fractures through the host rock and the surrounding country rocks. The driving mechanism for injection into the surrounding rock may be deformation or settling forces which push the liquid metal sulfide into new open spaces. Of the total identified world nickel resources, 30% to 40% has been estimated to be associated with sulfide ores (Table 2-5, Figure 2-1). However, in many places sulfide ores are the predominant economic sources of world nickel, accounting for two-thirds of the nickel that comes on the market every year. Beside Canada and the Soviet Union economic nickel sulfide deposits occur in Australia and Finland; non-economic deposits are known in southeast Alaska, Minnesota (Duluth Complex) and in parts of Greece and South Africa.

TABLE 2-5

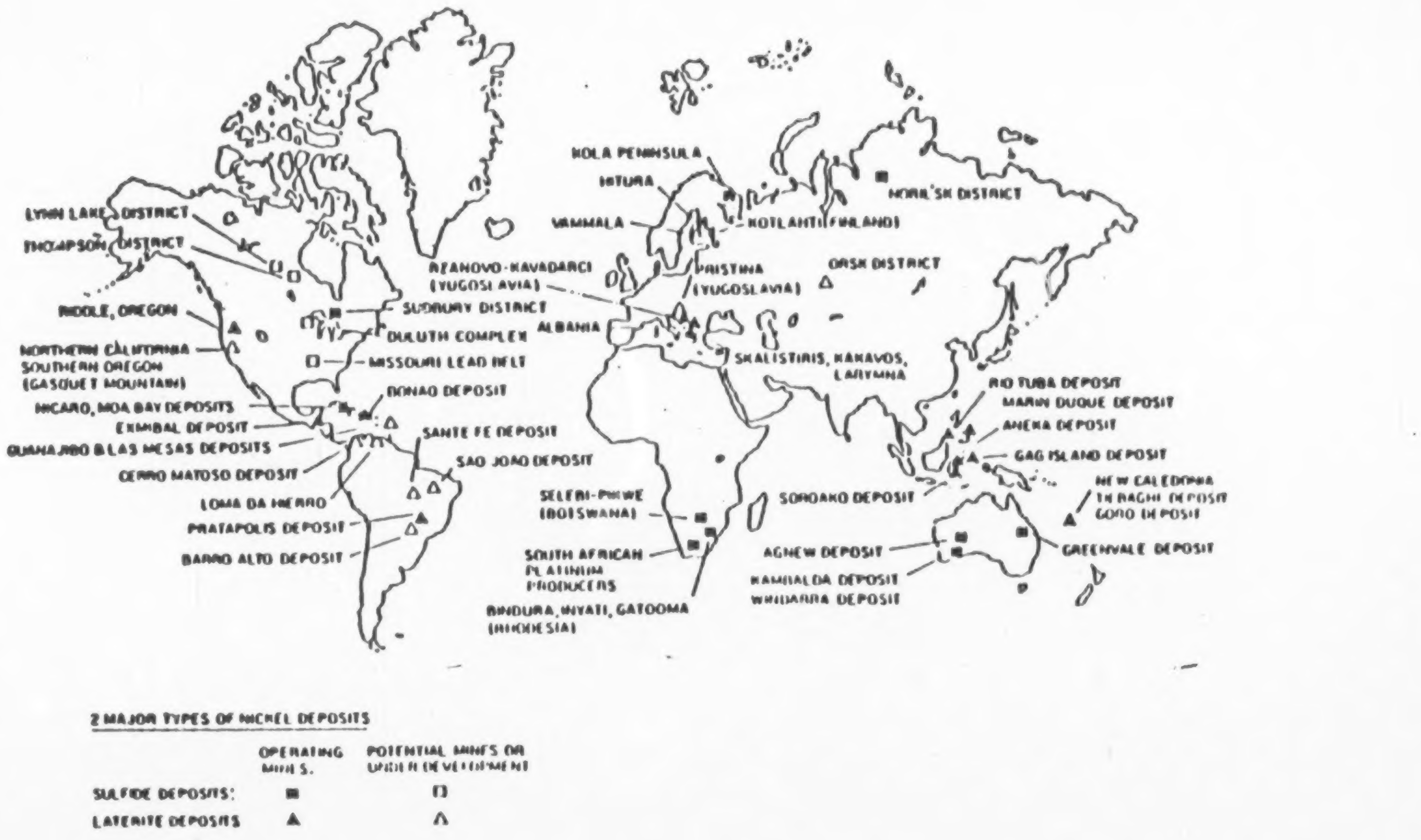
USBM AND UNITED NATIONS ESTIMATES OF WORLD RESERVES AND RESOURCES OF NICKEL, 1979
(000 tons metal)

Region and Country	Reserves		Resources			Total
	USBM 1979	United Nations 1979	USBM 1979	United Nations 1979	USBM 1979	United Nations 1979
NORTH AMERICA						
United States	200	220	14,900	331	15,100	551
Canada	8,700	8,375	12,500	8,265	21,200	16,640
TOTAL	8,900	8,595	27,400	8,596	36,300	17,191
SOUTH AND CENTRAL AMERICA						
Brazil	460	2,149	3,640	992	4,100	3,141
Columbia	900	1,543	600	220	1,500	1,763
Cuba	3,400	3,747	14,200	18,293	17,600	22,040
Dominican Republic	1,100	1,102	100	771	1,200	1,873
Guatemala	300	551	900	1,102	1,200	1,653
Puerto Rico		--	900	1,102	900	1,102
Venezuela		661	700	496	700	1,157
TOTAL	6,160	9,753	21,040	22,976	27,200	32,729
EUROPE-CHINA						
Albania		220		331		551
China		827		NA		827
Finland		882		--		882
Greece		2,204		2,535		4,739
USSR	8,100	9,918	13,200	5,510	21,300	15,428
Yugoslavia		882		661		1,543
TOTAL	8,100	14,933	13,200	9,037	21,300	23,970
OCEANIA						
Australia	5,600	2,755	3,200	7,163	8,800	9,918
Indonesia	7,800	9,698	55,000	5,510	62,800	15,208
New Caledonia	15,000	14,877	31,800	33,060	46,000	47,937
Philippines	5,700	7,163	10,600	33,060	16,300	40,223
Other		551		3,747		4,298
TOTAL	34,100	35,044	99,800	82,540	133,900	117,584
AFRICA						
Botswana		551		--		551
Republic of South Africa		882		3,802		4,684
Rhodesia		661		5,675		6,337
Other (Burundi-Madagascar)		331		5,290		5,620
TOTAL	2,300	2,425	6,700	14,767	9,000	17,192
WORLD TOTAL	59,560	70,750	168,140	137,916	227,700	208,666
GEOPOLITICAL SUB-UNITS						
United States	200	220	14,900	300	15,100	500
Central Economy Countries	11,500	15,594	27,400	24,795	38,900	40,389
World exclusive of U.S.	59,360	70,530	153,240	137,616	212,600	208,166
World exclusive of Central Economy Countries	48,060	55,156	140,740	113,121	188,800	168,277

Source: USBM, Mineral Commodity Profile, Nickel - 1979; United Nations, 1979.

FIGURE 2-1

IMPORTANT NICKEL RESERVES AND RESOURCES


LYNN LAKE DISTRICT
THOMPSON DISTRICT
RIDDLE, OREGON
NORTHERN CALIFORNIA
SOUTHERN OREGON
(GASQUET MOUNTAIN)
NICARO, MOA BAY DEPOSITS
EXMIBAL DEPOSIT
GUANAJIBO & LAS MESAS DEPOSITS
CERRO MATOSO DEPOSIT
LOMA DA HERRO
PRATAPOLIS DEPOSIT
BARRO ALTO DEPOSIT
RZANOVO-KAVADARCI
(YUGOSLAVIA)
SUDBURY DISTRICT
DULUTH COMPLEX
MISSOURI LEAD BELT
BONAO DEPOSIT
SANTE FE DEPOSIT
SAO JOAO DEPOSIT
KOLA PENINSULA
HITURA
VAMMALA
KOTLAHTI (FINLAND)
PRISTINA
(YUGOSLAVIA)
ALBANIA
SKALISTIRIS, KAKAVOS,
LARYMNA
ORSK DISTRICT
NORIL'SK DISTRICT
SELEBI-PHIKWE
(BOTSWANA)
SOUTH AFRICAN
PLATINUM
PRODUCERS
BINDURA, INYATI, GATOOMA
(RHODESIA)
RIO TUBA DEPOSIT
MARIN DUQUE DEPOSIT
ANEKA DEPOSIT
GAG ISLAND DEPOSIT
SOROAKO DEPOSIT
NEW CALEDONIA
THIBAGHI DEPOSIT
GORO DEPOSIT
AGNEW DEPOSIT
KAMBALDA DEPOSIT
WINDARRA DEPOSIT
GREENVALE DEPOSIT
2 MAJOR TYPES OF NICKEL DEPOSITS
OPERATING MINES.
POTENTIAL MINES OR UNDER DEVELOPMENT
SULFIDE DEPOSITS:
LATERITE DEPOSITS

Source: Dames & Moore, 1980.

Laterite Deposits

The other major type of economic nickel occurrence is in nickeliferous laterites. As discussed in the cobalt section, these laterites occur on a host of mafic and ultramafic materials, particularly dunites and peridotites. These ores develop in areas of current or past subtropical weathering, where residual soils are concentrated in nickeliferous materials. Distribution of nickel in the soil usually follows that of iron in the higher part of the soil profile; therefore, the ores are often limonitic and of uniform grade. In the lower part of the soil profile, nickel is associated with magnesium silicates of the host rock. Nickel concentrations in the limonitic near-surface ores often range from between 0.82% to 1.1%; in the lower part of the soil profile magnesium silicates dominate the mineralogy, and nickel concentrations are generally above 1%; it may even exceed 4% to 5% in some instances. Average ore grades, therefore, reflect a combination of the mining mixture, the relative proportion of limonitic to silicate ore within each local mining site. Nickel ore grades are usually much more uniform than those of cobalt in the same deposit (Foose, 1982).

The chemistry of these deposits is dramatically controlled by the interaction of the rock type chemistry, climate and topography. The intensity of chemical weathering plays an important role in the distribution of nickel, and cobalt as well, and a residual accumulation of ore materials is primarily a function of microclimate and topographic interaction at the site. It is, therefore, expectable that in areas of primary nickel-bearing rock, in the form of .paultramafics, slight differences in the character in each potential mining site may play a significant role in affecting the ore grade of that site.

On a worldwide basis there are two zones of nickel occurrences within a laterite profile: the limonitic occurrence and the nickel silicate occurrence. Each can also be segregated on the basis of original rock type on which it forms. Limonitic ores are more clearly developed upon a serpentine base, where little, if any, of the original peridotite parent is preserved. In contrast, the nickel silicate laterites are more likely to occur when formed over peridotite or pyroxinite rocks, where pyroxenes are the main rock-forming mineral.

The major nickel-bearing lateritic ores worldwide occur in Central America, Indonesia, New Caladonia, the Philippines, and in northern California and southern Oregon. These ores comprise 50% to 60% of the world resource base for nickel (Table 2-5).

CHROMIUM

Description and Uses

In its pure form chromium is steel-gray with a dull metallic luster. It has an atomic number of 24, an atomic weight of 52, a melting point of 1857°C, and a specific gravity of 7.2. In the transportation, manufacturing, and energy industries, chromium is an essential, although minor, component of material demand. Its most important use is as an alloying agent in stainless steel, an area in which substitute materials or use reduction can effectively decrease chromium demand. Chromium is also used in non-stainless steel alloys, and in this form, it is largely an irreplaceable component. Chromium is also irreplaceable in high-temperature-resisting super alloys used in aerospace and petroleum cracking.

Types of Deposits

Like nickel, the chromium deposits of the world are associated with mafic and ultramafic intrusives, primarily anorthositic and peridotite rocks. There are two major primary geologic types of ore, the stratiform/layered and podiform or alpine deposits. A minor occurrence in laterite deposits is also potentially producable.

Stratiform/Layered Deposits

Stratiform or layered deposits are characterized by uniformity and consistent position of chromite banding in the host complex. That complex is usually a layered mafic or ultramafic rock, and the chromite zone is found in the lower part of the ultramafic. Chromite horizons usually are found separate from zones of copper/nickel or platinum concentration within these host rocks. The occurrence of chromite in the lower parts of these layered intrusives and the stratiform character of the intrusive itself appears to suggest fractional crystallization and settling as a major process related to the occurrences of ores.

There are few great examples of stratiform chromite deposits in the world; the most important are the Bushveld Complex of South Africa, the Great Dike of Zimbabwe, the Stillwater Complex of Montana and the Bull River Complex of Manitoba, Canada. Although there are similarities between all four of these stratiform rock bodies, they are not exactly correlative throughout their known exposure. The overwhelming majority of the world's chromite resources exist within these strataform deposits (Table 2-6, Figure 2-2) and make up the largest sources of potential supply well into the next century. The stratiform bodies generally yield chromites of the high-iron variety and are, therefore, used primarily as chemical grade ores.

TABLE 2-6

CHARACTERISTICS OF THE MAJOR KNOWN STRATIFORM CHROMITE DEPOSITS

Location	Estimated Reserves[a] (000 Tons) of ore)	Estimated Current (1979) Production[b]	Principal Product Grade[c]	Typical Thickness of Seam	Mining Method
Bushveld area of South Africa	2,500,000	3,500[d]	chemical	Eastern Belt 40-44 inches Western Belt less than 12 inches	Underground
Great Dyke area of Zimbabwe	1,100,000	660[e]	metallurgical and chemical	3 to 7 inches	Underground
Kemi area of Finland	28,000	475[f]	high-iron 27.4% Cr_2O_3	---	Open Pit
Stillwater Complex Montana	subeconomic resource	-0-	high-iron low Cr grade 22.5% Cr_2O_3	2 to 16 feet	Abandoned underground Workings
Bird River area of Manitoba	subeconomic resource	-0-	high-iron low Cr grade 25% Cr_2O_3 or less	---	No mining operation

---

[a] Unpublished USBM estimates.

[b] USBM 1980 Mineral Commodity Summaries; gross weight of shipping ore.

[c] USBM Mineral Commodity Profile: Chromium-1977.

[d] Estimated 1979 production of South Africa.

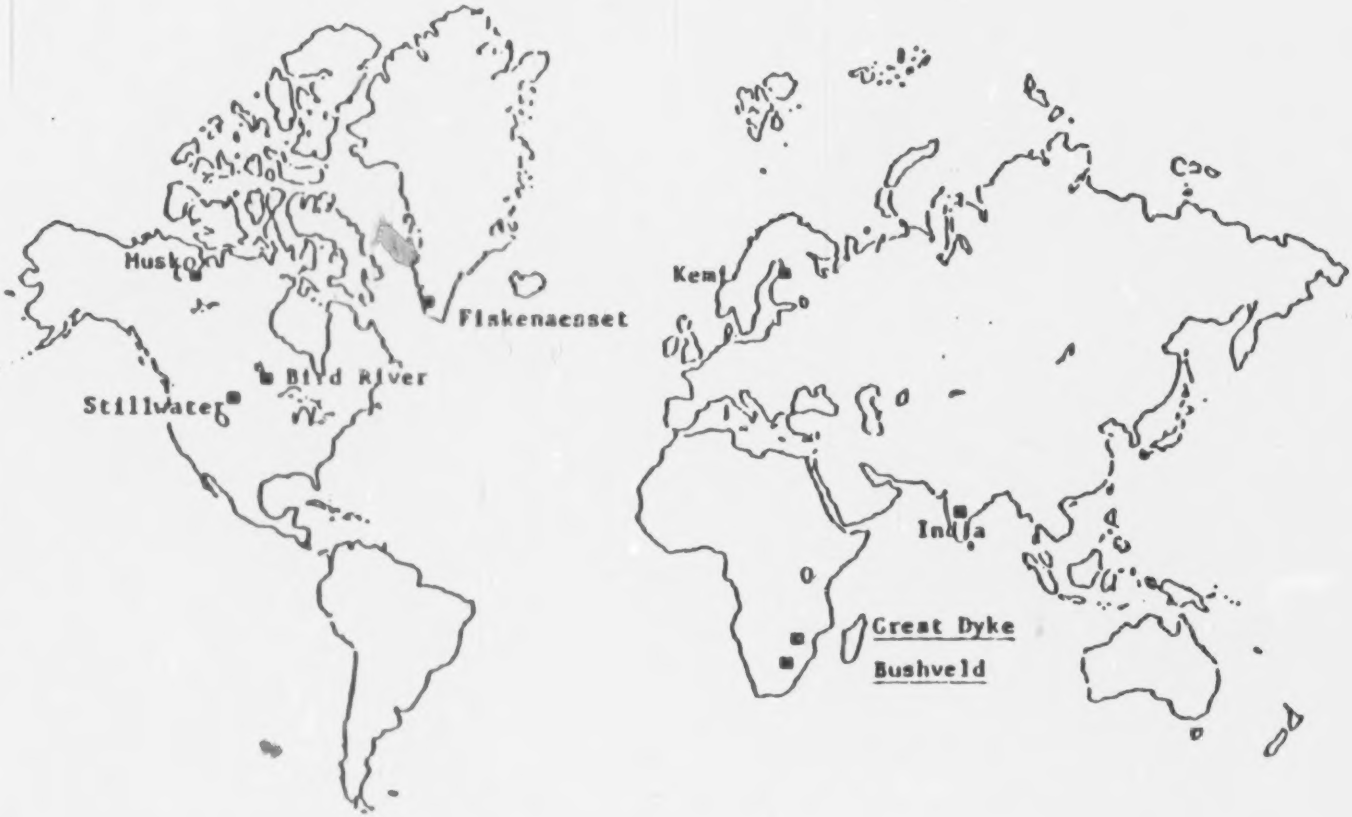
[e] Estimated 1979 production of Zimbabwe.

[f] 1979 Engineering and Mining Journal International Directory.

Source: Dames & Moore, 1980.

FIGURE 2-2

IMPORTANT CHROMITE RESERVES AND RESOURCES IN STRATIFORM DEPOSITS


Muskox
Fiskenaesset
Kemi
Bird River
Stillwater
India
Great Dyke
Bushveld

Source: Dames & Moore, 1980.

Podiform Deposits

Podiform (alpine) deposits of chromite occur in peridotitic and mafic terranes which are usually associated with subduction zones and accretionary slabs on active continental margins. Deposits are commonly tabular to irregular in shape and oriented parallel to the major structural trend of the enclosing host rocks. Margins of deposits are usually well defined but may also be, in part, diffuse and irregular at any particular location. In contrast to the strataform host rocks, these deposits appear to be related to the intrusion of small basic and ultrabasic magmas derived from the subducted plate and perhaps injected into the crust as a semisolid material, undergoing hydrothermal alteration along the way. The strataform ores, on the other hand, result either from massive melting of the mantle, in essentially a nontectonic environment along zones of major pressure release, or the partial melting of the mantle as a result of meteorite impact on the continental crust.

The podiform deposits are usually small in size, rarely exceeding 300-500 ft in the longest dimension. Therefore, they represent small-scale mining opportunities should they ever again be commercial in the U.S. They represent less than 5% of the world's known chromite resourses, although approximately 50% of the 1979 world production of chromite was derived from these podiform ore types (Table 2-7, Figure 2-3). The occurrence of podiform bodies in faulted mountain melts and accretionary sedimentary prisms associated with once, or presently, active subduction margins are known in the Ural Mountains of the USSR, the alpine mountain system of Turkey, the Zambouly Mountains of the Philippines and in the Coast Ranges of Oregon and California. These deposits are the primary source of the world's metallurgical-grade high

TABLE 2-7

ESTIMATED WORLD CHROMIUM RESOURCES BY DEPOSIT TYPE AND PRODUCT GRADE[a]

(in percent)

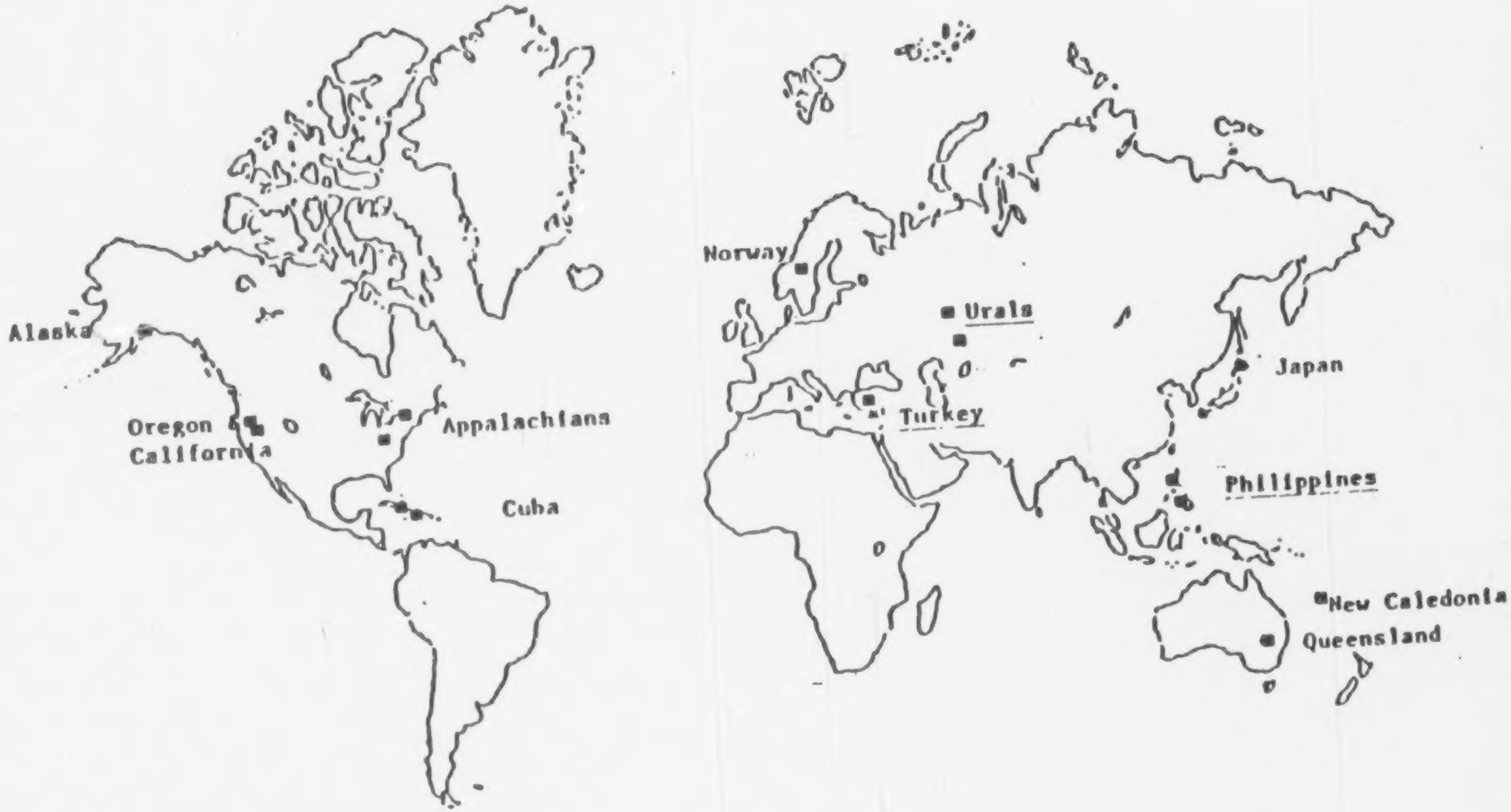
Grade	Stratiform Deposits	Podiform Deposits
High Chromium	30	78
High Iron	70	5
High Aluminum	--	17
	100	100

Source: Dames & Moore, 1980.

[a] Resources contained in podiform deposits account for only 1 percent of total world resources. About 99 percent is contained in stratiform deposits.

FIGURE 2-3

IMPORTANT CHROMITE RESERVES AND RESOURCES IN PODIFORM DEPOSITS


Alaska
Oregon
California
Appalachians
Cuba
Norway
Urals
Turkey
Japan
Philippines
New Caledonia
Queensland

Source: Dames & Moore, 1980.

chromium ores and, therefore, are more valuable in the world market. They are also virtually the only source of refactory grade chromite.

Laterite Deposits

Lateritic or residual chromites derived from the tropical weathering of appropriate rock types such as peridotites, pyroxenites and serpentines, can form chromites of sufficient grade and tonnage, with or without nickel and cobalt, to be mineable someday. Like the nickel laterites these ores occur as residual chromite derived from intense chemical weathering of the underlying rocks. During the weathering process, the small amount of chromite that might be originally present in the host rock is concentrated in the residual soil blanket, thereby upgrading the chromite concentration by a factor of 5-10 times, creating nearly economic concentrations of chromium. Lateritic chromium occurrences have been noted in California, Oregon, Cuba, the Philippines, Africa and the Ural Mountains of the USSR. World occurrences of these chromite-bearing laterites may contain a significant amount of chrome materials; they are not, however, usually of high enough concentration to be the primary ores of any major mining operation.

MANGANESE

Description and Uses

Manganese is much like chrome in its manner of use, though much more of it is consumed than of chrome, nickel, or cobalt. Manganese oxide and ferromanganese consumption is tied to the making of steel, and, therefore, the major consumers of manganese are the large steel manufacturers of the world. Approximately 10% of the world supply of manganese is consumed outside of steel making, mainly in the manufacture of dry cell batteries. The value of manganese derives mainly from the beneficial effects it has on iron and steel, particularly in the process of deoxidation, desulfurization, inhibiting grain boundary carbide formation and providing increased hardness and hardening ability when it alloys with iron to make either iron castings or steel. Manganese cannot alloy with other metals; manganese oxide can be a depolarizer for dry cell batteries or an oxidant in the production of a variety of chemicals.

Types of Manganese-Bearing Deposits

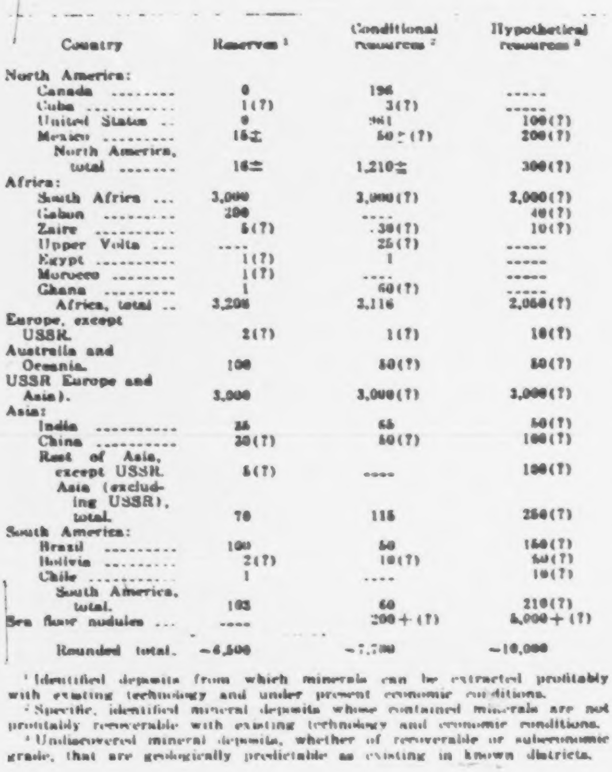
The potential economic manganese resources of the world can be classified in four major geologic types: sedimentary deposits, hypogene or exhalative deposits, manganese ores in metamorphic rocks, and surficial or residual-lateritic deposits. This classification implies a wide range of geologic conditions for the concentration of manganese in economic proportions in the earth's crust. The countries in which the manganese resources occur, and estimated reserves and resources are given in Table 2-8.

Sedimentary Deposits

The largest occurrences worldwide for potentially economic manganese deposits are sedimentary in origin. Deposits are of two types: those that are volcanogenic-sedimentary and those that are nonvolcanogenic-sedimentary. The distinction hinges on whether manganese is derived from volcanic/sedimentary activity or is derived from the weathering of crustal rocks.

Sedimentary deposits are found associated with different lithologic types, ranging from clastic to chemical sediments. Tectonic frameworks for ore deposition, ranging from active mobile continental margins to stable platform environments can be involved in ore formation through weathering and

TABLE 2-8
World Manganese ore resources, in millions of tons



Country	Reserves [1]	Conditional resources [2]	Hypothetical resources [3]
North America:			
Canada	0	196	-----
Cuba	1(?)	3(?)	-----
United States	0	961	100(?)
Mexico	15±	50±(?)	200(?)
North America, total	16±	1,210±	300(?)
Africa:			
South Africa	3,000	3,000(?)	2,000(?)
Gabon	200	----	40(?)
Zaire	5(?)	30(?)	10(?)
Upper Volta	----	25(?)	-----
Egypt	1(?)	1	-----
Morocco	1(?)	----	-----
Ghana	1	60(?)	-----
Africa, total	3,208	3,116	2,050(?)
Europe, except USSR.	2(?)	1(?)	10(?)
Australia and Oceania.	100	50(?)	50(?)
USSR Europe and Asia).	3,000	3,000(?)	3,000(?)
Asia:			
India	35	65	50(?)
China	30(?)	50(?)	100(?)
Rest of Asia, except USSR.	5(?)	----	100(?)
Asia (excluding USSR), total.	70	115	250(?)
South America:			
Brazil	100	50	150(?)
Bolivia	2(?)	10(?)	50(?)
Chile	1	----	10(?)
South America, total.	103	60	210(?)
Sea floor nodules	----	200+(?)	5,000+(?)
Rounded total.	~6,500	~7,700	~10,000

[1] Identified deposits from which minerals can be extracted profitably with existing technology and under present economic conditions.

[2] Specific, identified mineral deposits whose contained minerals are not profitably recoverable with existing technology and economic conditions.

[3] Undiscovered mineral deposits, whether of recoverable or subeconomic grade, that are geologically predictable as existing in known districts.

Source:

transport of manganese to sedimentary basins. Ores range in age from Precambrian to Cenozoic, with by far the major accumulation of terrestrial manganese resources in rocks of Cenozoic age.

Volcanogenic-sedimentary deposits can be directly associated with volcanic sources areas through volcanic exhalations of manganese-rich materials or from the weathering of manganese-rich volcanic rocks. Sedimentary manganese beds are found in close association with volcanic rocks throughout the geologic record and may also be found with hydrothermal deposits of both sedimentary and hypergene type. The deposits of Morocco, Sweden, Ghana and the Olympic Peninsula of the United States are examples of the volcanogenic-sedimentary manganese environments.

The nonvolcanogenic-sedimentary ores are essentially those in which volcanic sources of manganese, in either space or time, have not been identified and in which manganese ores appear to have been derived from the weathering of a continental land mass, transportation of the manganese to basins in deposition, and the accumulation of primary manganese oxide and carbonate mineralogy in association with clastic sedimentation. These ores are known to grade into manganiferous limestones and dolomites, and important manganese deposits such as Chiatura and Nickelpol of the USSR, and the Brazilian and Indian ores are composed of this type. Sedimentary derived manganese deposits associated with metasedimentary iron formations, all of probable Precambrian age, make up an important resource base from South Africa, Brazil and the USSR. These manganese ores are probably nonvolcanogenic-sedimentary deposits originally formed in close association with sedimentary iron formation. Subsequent burial and metamorphism have altered, to some degree, the mineralogy of the ore beds, but they provide significant continuity of thickness and strike-length that make them important sources of manganese on a worldwide basis.

Hypogene or Exhalative Deposits

These manganese deposits develop when circulating thermal waters dissolve varying amounts of mineral components from either crystallizing igneous rocks or by circulation through sedimentary and metamorpkic rocks. These mineral-bearing waters rise toward the surface, decrease in temperature and pressure and unload or drop their chemical content in a variety of structures, including fractures along which veins may form, or as replacements for carbonate and other rocks. Hypogene manganese deposits derived from hydrothermal waters are commonly found associated with other ores, including base metal sulfides and precious metals.

Important examples of the hypogene/hydrothermal deposits occur in a variety of localities in the United States, the USSR, Morocco and Japan. The most notable U.S. examples include the Philipsburg and Butte districts of Montana and the San Juan region of Colorado. The ores in these environments are often high grade, but underground mining costs are usually prohibitive and unless associated with other ores of high demand they are uneconomic to mine for the manganese content alone. Small bodies of hypogene manganese ores are known from all over the world but together make up only a small part of the resource base.

Metamorphic deposits

Primary sedimentary manganese ore deposits can be subjected to dynamic and thermal metamorphism, which changes the original deposit in mineralogy and texture. Regional metamorphism affecting manganese-bearing material rarely changes the grade of ore, but can account for a mineralogic transformation that can make the ores less valuable. In particular, the conversion of manganese oxides and carbonates to manganese silicates has a negative impact on the economic viability of the deposit, producing resources that are difficult to process with current technology. The ores of northern Maine (Aroostoock County) and the north Cuyuna ores of Minnesota are of this type.

Surficial/Residual Deposits

Supergene, residual enrichment of low grade manganese-bearing material can provide high grade manganese resources. The concentration process for these ores is similar to that which forms lateritic iron and nickel deposits. Manganese remains behind in weathering and is thereby enriched in the material of the lateritic soils. These ores are derived from either manganese carbonates or oxides, or perhaps even manganese silicate rocks. The ore bodies are usually of small size, irregular in grade and composition and not as important as primary sources of manganese ores described above. There are, however, manganese deposits in Brazil, Mexico, Gabon and Ghana, as well as occurrences in Australia and India, that are of sufficient size and grade to have been mined successfully in the past. Worldwide, these residual ores make up a small but important part of the economic resource base.

PLATINUM

Description and Uses

Platinum is a precious metal found in its elemental state along with other platinum-group metals and produced as either a byproduct in the refining of copper and other ores, or from placer deposits as the primary commodity. Platinum values are derived primarily through its use in the automotive industry (in catalytic converters) and also in electrical, chemical, dental, and jewelry consumption. Chemical and petroleum refining industries use platinum-group metals mainly as a catalyst. Potential substitutes for platinum in electrical uses are gold, silver and tungsten, titanium is a substitute in catalytic uses, and gold and silver proxy in dental and jewelry use. The continued use of platinum group metals in catalytic converters by the automobile industry depends primarily on the evolution of converter technology and the status of emission control system requirements for future automobiles.

Types of Platinum-Bearing Deposits

Together with chromium and nickel, world primary resources of platinum and platinum-group metals are restricted to the major basic and ultrabasic layered intrusive masses known in continental terrains. These large intrusives like the Bushveld, the Great Dike, Stillwater, Duluth Complex and others in the USSR, Australia, and elsewhere, provide the major source of platinum and platinum-group metals from primary mining environments (Table 2-9). In some cases, such as in the Stillwater, these primary stratiform ore bodies can be mined primarily for their platinum content. Here the platinum mineralization is separated from, and independent of, the other potential ore-bearing phases of the stratiform rocks, namely chromite, nickel, and copper.

Platinum in igneous stratiform environments clearly makes up the only known important primary concentration of platinum in the earth. Chemical and physical weathering can segregate platinum minerals into stream and beach placer resources, where secondary concentration can provide ore grades

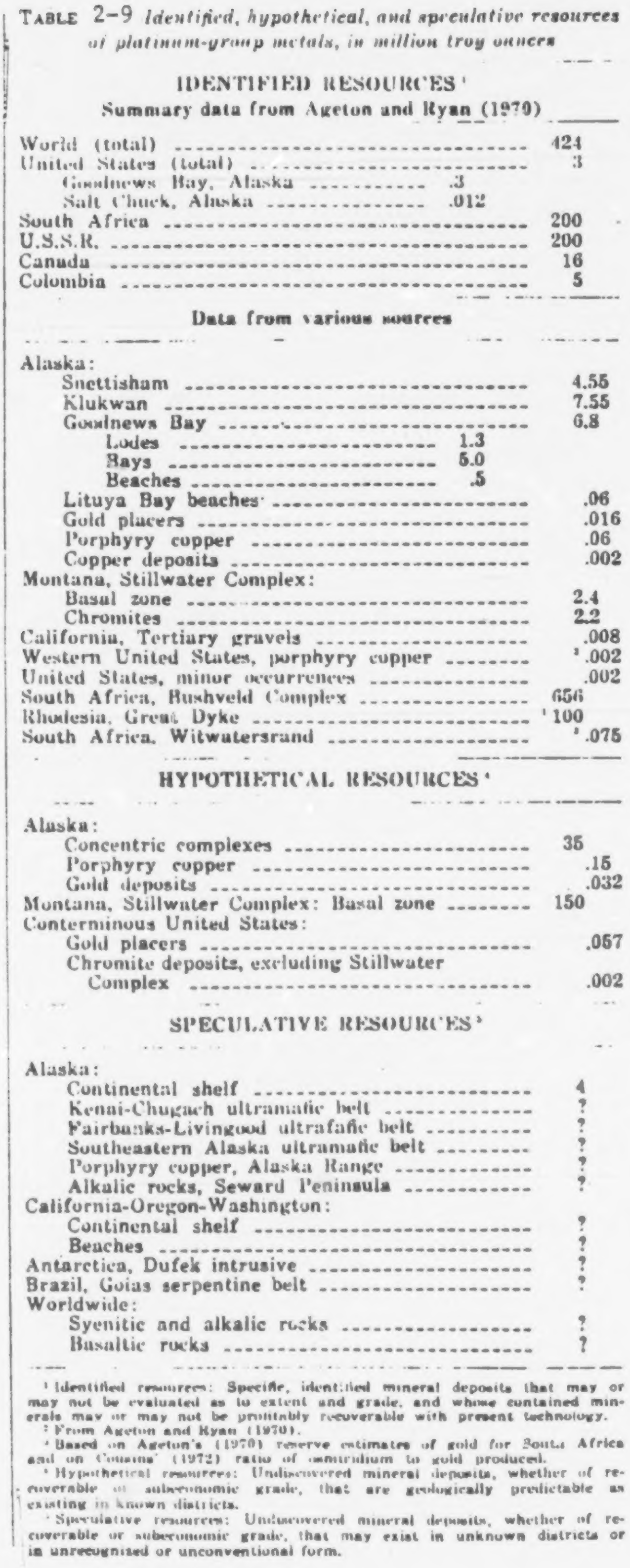


TABLE 2-9 *Identified, hypothetical, and speculative resources of platinum-group metals, in million troy ounces*

IDENTIFIED RESOURCES[1]		
Summary data from Ageton and Ryan (1970)		
World (total)		424
United States (total)		3
Goodnews Bay, Alaska	.3	
Salt Chuck, Alaska	.012	
South Africa		200
U.S.S.R.		200
Canada		16
Colombia		5
Data from various sources		
Alaska:		
Snettisham		4.55
Klukwan		7.55
Goodnews Bay		6.8
Lodes	1.3	
Bays	5.0	
Beaches	.5	
Lituya Bay beaches		.06
Gold placers		.016
Porphyry copper		.06
Copper deposits		.002
Montana, Stillwater Complex:		
Basal zone		2.4
Chromites		2.2
California, Tertiary gravels		.008
Western United States, porphyry copper		[2] .002
United States, minor occurrences		.002
South Africa, Bushveld Complex	656	
Rhodesia, Great Dyke	[2] 100	
South Africa, Witwatersrand		[3] .075
HYPOTHETICAL RESOURCES[4]		
Alaska:		
Concentric complexes	35	
Porphyry copper		.15
Gold deposits		.032
Montana, Stillwater Complex: Basal zone	150	
Conterminous United States:		
Gold placers		.057
Chromite deposits, excluding Stillwater Complex		.002
SPECULATIVE RESOURCES[5]		
Alaska:		
Continental shelf		4
Kenai-Chugach ultramafic belt		?
Fairbanks-Livingood ultrafafic belt		?
Southeastern Alaska ultramafic belt		?
Porphyry copper, Alaska Range		?
Alkalic rocks, Seward Peninsula		?
California-Oregon-Washington:		
Continental shelf		?
Beaches		?
Antarctica, Dufek intrusive		?
Brazil, Goias serpentine belt		?
Worldwide:		
Syenitic and alkalic rocks		?
Basaltic rocks		?

---

[1] Identified resources: Specific, identified mineral deposits that may or may not be evaluated as to extent and grade, and whose contained minerals may or may not be profitably recoverable with present technology.

[2] From Ageton and Ryan (1970).

[3] Based on Ageton's (1970) reserve estimates of gold for South Africa and on Cousins' (1972) ratio of osmiridium to gold produced.

[4] Hypothetical resources: Undiscovered mineral deposits, whether of recoverable or subeconomic grade, that are geologically predictable as existing in known districts.

[5] Speculative resources: Undiscovered mineral deposits, whether of recoverable or subeconomic grade, that may exist in unknown districts or in unrecognized or unconventional form.

Source: USGS, 1973, p. 541.

amenable to mining. Placer ores are, therefore, the only alternative to the primary strataform deposits, which today make up the bulk of platinum supply.

A couple of world class stratiform bodies are capable of yielding important platinum resources within the United States. These are the Stillwater Complex of Montana and the Duluth Gabbro Complex of Minnesota. Significant amounts of platinum may be eventually recovered from both bodies. There may be, however, some as yet undiscovered platinum resource in stratiform and other ultrabasic rocks of the type described above, in Alaska. Inasmuch as recent Alaskan platinum resource discoveries are primarily placer type, the parent material may be basic and ultrabasic rocks of the Stillwater or Duluth type. These rocks, however, may be buried under snow and ice most of the year and only accessible to year-round exploration and development by both remote sensing and creative mining technologies.

Chapter 2

BIBLIOGRAPHY

Brobst, D.A., and Pratt, W.P., editors, 1973, U.S. Mineral Resources, Geological Survey Professional Paper 820. Washington, D.C.: U.S. Department of the Interior.

Dames & Moore Minerals and Resources Economic Group, 1980, Cobalt, Chromium, and Nickel Supply, Demand and Strategic Importance. San Francisco: Dames & Moore.

Foose, M.P., 1982, Geology of Important Cobalt-bearing Deposits and Implications Regarding Future Development of Domestic Cobalt Resources. In Congressional Hearings on Strategic Minerals, 1981, Appendix 1.

U.S. Bureau of Mines, 1983, Mineral Commodity Profiles: Nickel. Washington, D.C.: U.S. Department of the Interior.

U.S. Bureau of Mines, 1982, Cobalt Availability--Domestic. A Minerals Availability System Appraisal (RI 8848; by G.R. Peterson, D.I. Bleiwas, and P.R. Thomas) Washington, D.C.: U.S. Department of the Interior.

Chapter III

ESTIMATES OF THE METAL RESOURCE BASE

Any attempt to provide numerical estimates of a total resource base potentially extractable within a 20-25 year time frame is fraught with a variety of dangers, both real and theoretical. In particular, the estimates are based primarily on the deposits already discovered and the resource data base associated with those deposits. The quality of that data base varies dramatically from place to place. In addition resources in the hypothetical and speculative categories are based primarily on geological and geochemical inference and the application of geostatistic elements in known resource environments to those as yet unexplored.

For ore deposits that have already been identifed, the quality of the resource estimates varies from place to place as a function as the amount of exploration and development, the known size of the deposit, the inference on ore grade continuity, and to some extent the assumptions about future technological innovation and market structure. It is not surprising, therefore, that only a few major mineral deposits have been completely evaluated for their reserve base, and even these lack good information on potentially recoverable inferred ore.

Chromium, platinum and nickel ores are usually associated primarily with layered ultrabasic and mafic intrusions. Because the ore grade is continuous, the layering of chromite-, platinum- and nickel-bearing sulfides within the stratiform bodies is pronounced, and the bodies themselves are relatively large, it is esaier to estimate the resource base for these three commodities than for cobalt and manganese.

CHROMIUM

The estimated worldwide chromium reserve base is approximately 3.7 billion tons (Table 3-1), about 2.5 billion tons of which is in the Bushveld Complex of the Republic of South Africa, and most of the rest in the Great Dike of Zimbabwe. Another 30 billion tons of chromite have been identified in the world resource base and probably exists, but is not now amenable to the economics of mining and market. Again, almost all of this resource base is in the Bushveld and the Great Dike. The total reserve and resource estimates of chromite is, therefore, enormous, and more than 95% of it is in the two world class deposits of the layered stratiform type.

Other chromite reserves and resources of reasonably significant amounts are found in the Soviet Union, where current assessments suggest reserves and resources in excess of 120 million tons of chromite; in India with 90 million tons of resources and reserves; and in Finland, which contains 80 million tons. Small amounts of chromite resources are known to occur in the Philippines, Turkey, Cuba, Madagascar, Brazil and Albania.

The podiforn/disseminated chromites characteristic of subduction regimes contain only small amounts of the reserves and resource base, the total of which cannot be estimated accurately at this time. It is clear, however, that it is only a minor fraction of the total world resource and reserve base.

The chromite resources themselves portend the future mining and economic relationship of this valuable material. Specifically, the two major world deposits could supply world needs for many decades to come. Smaller amounts of chromite are available from a variety of podiform sources around the world, and can substitute on a short term basis for the chromite supply from South

TABLE 3-1

ESTIMATED WORLD CHROMIUM RESERVES AND RESOURCES BY COUNTRY AND REGION
(000 tons, gross weight of shipping ore)

Region/Country	Reserves		Resources		Primary Ore Grade
	Quantity	Percent	Quantity	Percent	
North and South America:					
United States	—	-	11,000	0.03	High-iron
Brazil	11,000	0.29	22,000	0.07	High-iron, high-chromium
Canada	—	-	11,000	0.03	High-iron
Cuba	3,300	0.09	11,000	0.03	High-aluminum
Greenland	—	-	11,000	0.03	High-iron
Total	14,300	0.38	66,000	0.20	
Asia and Oceania:					
India	60,000	1.60	28,000	0.09	High-chromium, high-iron
Iran	2,200	0.06	11,000	0.03	High-chromium
Philippines	3,300	0.09	22,000	0.07	High-aluminum
U.S.S.R.	17,000	0.45	100,000	0.31	High-chromium, high-aluminum
Total	82,500	2.20	161,000	0.50	
Europe:					
Albania	2,200	0.06	17,000	0.05	High-chromium
Finland	28,000	0.75	55,000	0.17	High-iron
Turkey	5,500	0.15	22,000	0.07	High-chromium
Total	35,700	0.95	94,000	0.29	
Africa:					
Madagascar	2,200	0.06	11,000	0.03	High-chromium
Zimbabwe	1,100,000	29.3	10,000,000	30.9	High-chromium high-iron
South Africa	2,500,000	66.6	22,000,000	68.0	High-iron, high-chromium
Total	3,602,200	95.9	32,011,000	99.0	
Others	22,000	0.59	11,000	0.03	
TOTAL	3,756,700	100.0	32,343,000	100.0	

Source: Dames & Moore, 1980, p. A-82, based on data from USBM and USGS, Pers. Conversation, 1980.

NOTE: Totals may not add to 100 percent because of rounding.

[a] Exclusive of reserves; also excludes any chromium in laterites.

Africa or Zimbabwe. It is, therefore, easy for these two countries to set world chromite prices and largely control the amount of chromite that reaches the international market. Disruption of supply from one or both of the African sources can be replaced for relatively short periods by other world sources or by domestic stockpiles; however, it appears impractical to have significant amounts of alternate supply on hand in the event of catastrophic supply interruptions from South Africa or Zimbabwe.

In addition, the easily expanded mining capacity in the two African states makes it even more difficult for new deposits to provide any economic competition, even with a modest growth in chromium use over the next 25 years. The maximum growth rate of 3% a year, predicted on international industrial growth rates, can be so easily accommodated in the currently known deposits of South Africa and Zimbabwe that price can be expected to rise, at the maximum, to just below the economic requirements for potential major competitors. It is clear, therefore, that for the next 25 years, the world price of delivered chromite will probably not rise much above $150 to $200 a ton in 1982 dollars.

NICKEL

Nickel resources are slightly more diverse, but again are related to the major large layered or stratiforms intrusives and nickel-rich laterite ores. When nickel occurs as disseminated, massive and vein sulfides, it is usually along with copper; nickel-rich laterites sometimes containing small amounts of cobalt and chromite. World nickel resources, therefore, can be defined as those contained in layered stratiform deposits, or in nickel laterites derived from the chemical weathering of basic and ultrabasic materials. Ore grades are comparable in both environments, and with adequate refining technology and capacity, surface nickel laterites should prove, in the long-term, to be competitive with sulfide ores because of the overall low cost of mining. Processing costs may drive the advantage the other way, however.

World nickel reserves are mainly contained within Canada, the USSR, the Philippines, New Caladonia, and Indonesia, together averaging about 70% of the known reserves. In addition, world nickel resources have been identified mainly in Canada, Cuba, New Caladonia, and the Philippines. Reserves on a world total basis have been estimated between 60-70 million tons of elemental nickel (see Table 2-5). More than half of that comes from the nickel laterite deposits of the Pacific rim, mainly, New Caladonia, Indonesia, and the Philippines. Nickel sulfide deposits associated with basic and ultrabasic intrusions make up the major resource of Canada (Sudbury, Ont.) and the USSR; reserves of both countries have been estimated by the U.S. Bureau of Mines and the United Nations to contain, on a combined basis, about 25% of world reserves. Smaller amounts of nickel, again primarily from latertitic ores, are derived from South and Central America.

The world resource base of nickel has been estimated to be between 140 to 170 million tons of the metal (see Table 2-5). Again upwards of 50% of the total is believed to be associated with the nickel laterites of the western Pacific area, while 10% to 15% is perhaps housed in the laterites of South and Central America. The stratiform and massive sulfide deposits of Canada, the U.S. and USSR share the remainder of the estimated resource base with each having approximately an equal share. Significant nickel resources are also associated with the stratiform layered intrusions of basic and ultrabasisc rocks in southern Africa. In North America, the major nickel resource is Sudbury district of Ontario; it is the largest nickel producing district in the world.

Potential economic nickel deposits of the United States are characterized by the nickeliferous laterites of California and Southern Oregon. The only major nickel producer in the country was at Hanna's Nickel Mountain, in Riddle, Oregon, which closed in April of 1982 as a result of modest increases in operating costs and the relatively low and falling demand for nickel. This deposit was capable of producing an important part of U.S. nickel output. At this time it appears to be a victim of relatively low nickel prices in a poor market as well as declining high grade reserves.

Like chromium, a few major ore bodies will provide most of the world demand for nickel for many years. World demand is only a very small fraction of total world potential supply from resource areas, and between South Africa, Canada, the USSR and the Pacific laterite states, the incentive is to keep market shares by keeping prices low. It is, therefore, unlikely that in the next 15 years nickel prices will escalate very much above the 1982 real price.

The best of the U.S. deposits, containing the richest nickel ores (Oregon/California laterites and Duluth Complex), will only be marginally competitive at times of shortage or high world demand. Significant calculation of the nickel price is unlikely and the international price of nickel will not rise much above a 2% to 4% annual rate over the next 15 years, about equal with inflation. Therefore, the best of the U.S. nickel deposits, Nickel Mountain at Riddle, Oregon, and the Duluth Gabbro ores, will essentially be noneconomic. These deposits need nickel prices at the 1982 base of from $6.00 to $8.00 a pound to come on line by 1995.

COBALT

Cobalt resources are indeed more diverse than either nickel or chromium with respect to the geologic occurrence and their distribution around the world. The major cobalt reserves are in Zambia and Zaire and consist of stratiform copper/cobalt sedimentary ores. However, significant resources also reside in U.S. bedded(?), elongate shear zone deposits in the central Idaho Blackbird district, with base metal sulfide ores in the east Missouri Lead Belt, as an accessory or trace element occurrence in nickeliferous laterites of California and Oregon, and in copper/nickel sulfide ores of the Duluth and Stillwater Complexes and the contact metamorphic sulfide ores of Pennsylvania.

Worldwide, significant resources are found in sulfide environments in Australia, Canada, Finland, Morocco, and the USSR, as well as in the laterites of the western Pacific and central and South America (Table 3-2). The ore grades in all of the above-mentioned environments are marginally to submarginally economic, except for the Zaire and Zambian deposits. Inasmuch as cobalt is now, and will continue to be, basically a byproduct of the mining of other primary ores, its relative economic position and anticipated availability are different than the other commodities of this study. Byproduct cobalt is recovered in Finland, Canada, Australia and the Philippines from sulfide and laterite ores.

World cobalt resource and reserve estimates have been a matter of some contention among professionals for a long time. The published world estimates by responsible resource organizations diverge by as much as a factor of two; however, the uncertainties are so great in the resource category that, in fact, the estimates may indeed be comparable. Estimates of world proven and

Table 3-2

ESTIMATES OF WORLD COBALT RESERVES AND RESOURCES
(Thousand Metric Tons)

	Reserves		Other Resources		Total Reserves and Resources	
	USBM*	BGR**	USBM*	BGR**	USBM*	BGR**
NONCOMMUNIST						
North America	27	220	998	430	1,025	650
Canada	27	220	227	30	254	250
United States	0	0	771	400	771	400
Europe	18	20	5	300	23	320
Finland	18	20	5	5	23	25
Greece	--	0	--	200	--	200
Spain	--	0	--	20	--	20
Yugoslavia	--	0	--	75	--	75
Africa	1,637	825	487	300	2,124	1,125
Botswana	27	25	5	5	32	30
Morocco	45	10	23	5	68	15
South Africa	23	--	5	--	28	--
Uganda	--	10	--	10	--	20
Zaire	1,179	450	227	230	1,406	680
Zambia	363	300	227	50	590	350
Zimbabwe	--	30	--	0	--	30
Asia/Oceania	317	1,555	567	2,740	884	4,295
Australia	45	135	249	165	294	300
Burma	--	0	--	15	--	15
India	--	45	--	0	--	45
Indonesia	--	565	--	300	--	865
New Caledonia	91	385	295	450	386	835
Papua/New Guinea	--	0	--	40	--	40
Philippines	181	425	23	1,750	204	2,175
Solomon Islands	--	0	--	20	--	20
Latin America	--	45	--	210	--	255
Brazil	--	30	--	70	--	100
Colombia	--	0	--	30	--	30
Dominican Republic	--	0	--	50	--	50
Guatemala	--	15	--	0	--	15
Puerto Rico	--	0	--	45	--	45
Venezuela	--	0	--	15	--	15
Subtotal	1,999	2,665	2,057	3,980	4,056	6,645

Table continued on following page.

Table 3-2 (cont'd.)

ESTIMATES OF WORLD COBALT RESERVES AND RESOURCES

	Reserves		Other Resources		Total Reserves and Resources	
	USBM*	BGR**	USBM*	BGR**	USBM*	BGR**
COMMUNIST						
Albania	--	0	--	5	--	5
China	--	0	--	20	--	20
Cuba	181	800	953	300	1,134	1,100
USSR	227	200	23	150	250	350
Subtotal	408	1,000	976	475	1,384	1,475
TOTAL	2,407	3,665	3,033	4,455	5,440	8,120

From: Charles River Assoc., 1982

SOURCES: * U.S. Bureau of Mines. 1980. Mineral Facts and Problems.

**Berlin: Deutsches Institut fur Wirtschaftsforschung; and Hanover: Bundesanstalt fur Geowissenschaften und Rohstoffe. 1978. Kobalt.

probable reserves vary between 1.5 million tons of cobalt metal to as much as 4 million tons. Approximately 90% of these reserves occur within the following nine countries: Zaire, Zambia, Indonesia, the Philippines, New Caladonia, the USSR, Cuba, Australia, and the United States. Nearly 50% of the proven and probable reserves of the world exist in just three countries, Zaire, Indonesia and Cuba (Table 3-2).

On the North American continent, cobalt reserves are localized in the mafic igneous rocks of the Sudbury district, and in the hydrothermal cobalt/silver ores of Cobalt, Ontario. In the United States the ores are not economic, and resources are divided between the nickel laterite deposits of southern Oregon and northern California, the hydrothermal vein or stratiform deposits of the Blackbird district of central Idaho, the lead sulfide ores of the Missouri Lead Belt, and the ultramafic intrusions of the Duluth and Stillwater? Complexes. The North American component of identified reserves is only a small fraction of the world total (Table 3-2).

With respect to the resource base, however, the Canadian deposits at Sudbury and the U.S. deposits of Blackbird, Idaho, the stratiform sulfides of the Duluth Complex and perhaps S.E. Alaska, and the cobaltiferous pyrites of the Missouri Lead Belt might account for as much as 10% to 12% of the total (Table 3-2). Of these, only the Blackbird ores could be mined for cobalt as the primary metal; the economic conditions under which mining could take place will be discussed in Chapter IX.

In summary, the world cobalt reserves and resources are localized primarily in one of three environments: (1) cobaltiferous sulfide ores in which the grade of ore may range as high as 5% cobalt, are usually in the 0.01% to 0.5% range and have near-term economic potential. The Sudbury and

Duluth ores are examples of this class. (2) The nickeliferous laterite deposits, which contain nearly three-quarters of the identified cobalt resources of the world. These cobalt ores are low grade, cannot be mined exclusively or primarily for cobalt in today's economic market, and range between .01% to 0.1% cobalt content. This large resource base, however, which may be as great as 3 to 5 million tons of cobalt, must be considered as an exploitable resource within a 25 year time frame. The richest of these cobalt-bearing, usually nickeliferous laterites, are in Cuba, the Philippines, and Indonesia. And (3) The sedimentary/stratiform sulfide ores of central Africa, which as yet have no important counterparts in similar geologic environments. These ores are extensive, high grade and are mined mainly for their cobalt and copper content; in addition, they dominate the world commercial market for cobalt.

MANGANESE

The world manganese reserve base has been estimated by the U.S. Bureau of Mines to be approximately 5.5 million tons of metal (Table 3-3). More than 4.5 million tons of that is equally distributed between the Republic of South Africa and the USSR. An order of magnitude less reserve is also known in Australia and Gabon and two orders less in Brazil, India, mainland China (Peoples Republic of China) and a few other free market countries.

Total resources of manganese are very large, irregularly distributed and not very well known. It is estimated that the USSR and the Republic of South Africa probably account for 75% of the identified world resource base; other countries with potentially large resources include Brazil, the Peoples Republic of China and Mexico (see Table 2-8). Within the United States the major identified manganese resources include the north Cuyuna Range of

Table 3-3A. WORLD MINE PRODUCTION AND RESERVE BASE OF MANGANESE

	Mine Production		Reserve Base*
	1980	1981 e/	
United States	-	-	-
Australia	2,162	2,200	330,000
Brazil	e/2,400	2,300	95,000
Gabon	2,366	1,800	159,000
India	1,814	1,800	50,000
South Africa, Republic of	6,278	6,600	2,200,000
Other Market Economy Countries	e/1,218	1,100	63,000
U.S.S.R.	11,300	11,400	2,400,000
China, Mainland	e/1,750	1,700	50,000
Other Central Economy Countries	e/141	100	25,000
World Total 8/	e/29,000	29,000	5,400,000

Source: USBM, 1982.

TABLE 3-3B. MANGANESE

MANGANESE

7. Government Programs:

Stockpile Status--11-30-81

Material	Goal	Total Inventory	Authorized For Disposal	Sales, 11 Months
Battery: Natural ore	62	185	45	10
Synthetic dioxide	25	3	-	-
Chemical ore	170	221	51	-
Metallurgical ore	2,700	2,409	-	-
Ferromanganese: High carbon	439	600	-	-
Medium carbon	-	29	-	-
Silicomanganese	-	24	-	-
Electrolytic metal	-	14	-	-

In addition to data shown, the stockpile contains 34,000 tons of natural battery ore and 961,000 tons of metallurgical ore, both of nonstockpile grade.

3

42 B

Minnesota, the Aroostook County manganese silicate deposits of Maine, the Chamberlain nodular ores of South Dakota, Artillery Peak, Arizona, and vein deposits in Pioche, Nevada, Butte, Montana, and Leadville, Colorado. The total resource base may be as much as 1 billion tons of contained manganese (Table 3-4). However, the average ore grade for these seven areas is only 7% Mn, containing about 65 million tons of manganese. The highest grade ores at Chamberlain are only half the world's commercial ore grade. The very low average grade and narrow range of all of the identified U.S. manganese continental resources makes it highly unlikely that they will be developed on a competitive basis over the next 15 to 25 years. Richer manganese deposits of both sedimentary and hydrothermal land-based types may be discovered in regions not thoroughly explored to date; however, until that time manganese mining in the United States, at best, will take the form of byproduct recovery.

Currently, the largest known concentration of manganese in the United States is in the carbonate nodules and enclosing manganiferous shale of the Chamberlain formation in South Dakota. The nodules lie near the surface within easy reach of mechanical stripping technology, and flat-lying manganiferous horizons may be continuous well beyond the present area of exploration. Total resource estimates of this deposit are probably only accurate within a factor of 2 or 3 (Table 3-4). However, for most of the manganese resources in the United States, (except for hydrothermal vein deposits of the western states) the size of the resource base is probably not better known than to within a factor of 5 to 10. Relatively little exploration and development information is available on manganese resources. What information we do have suggests only low-grade, noneconomic occurrences; those that could be mined commercially if all other higher grade resources and

TABLE 3-4—Major identified manganese resources[1] of the United States (USGS, 1982)

(In millions of long tons. Data slightly modified from Pavlides, (in Hewett and others, 1956, p. 214))

District	Raw material	Average grade (percent Mn)	Contained Mn
Cuyuna Range, Minn...........	450	5	22.5
Aroostook County, Maine......	280	9	25.2
Chamberlain, S. Dak..........	69	15.5	10.7
Artillery Peak, Ariz.........	156	4	6.2
Pioche, Nev..................	3	10	0.3
Leadville, Colo..............	3	15	0.5
Total..................	961	6.8	65.4

---

[1] Specific, identified mineral deposits that may or may not be evaluated as to extent and grade, and whose contained minerals may or may not be profitably recoverable with existing technology and economic conditions.

reserves were unavailable. United States resources, given the ore grades, could only be competitive on the world market with a price perhaps two to five times the current world price.

PLATINUM

The major primary platinum occurrences of the world are in ultramafic alpine intrusive complexes and in layered or stratiform basic rocks. In these environments, platinum occurs in background concentrations of a few to ten parts per billion. However, under some conditions platinum is concentrated to where mining might recover from a few tenths to half an ounce per ton. The secondary concentration of platinum in world class deposits is derived from the mechanical and chemical weathering of platinum-bearing host rocks with erosion and transport to stream and beach placers.

Identified platinum-group metal resources worldwide have been estimated at about 3.0 billion troy ounces (see Table 2-9). The majority of this comes from the Republic of South Africa, the USSR, and Zimbabwe. Estimates of platinum reserves are as high as 1.0 billion troy ounces with again the bulk of the ore in South Africa, Zimbabwe and the USSR (Table 3-5).

The identified primary sources of platinum in the world are clearly the major layered or stratiform basic to ultrabasic intrusions, including the Bushveld Complex of the Republic of South Africa and the Great Dike of Zimbabwe, the Duluth and Stillwater Complexes of the United States and possibly the Norsk'il Complex of the USSR; in addition, Alpine ultramafic zones of the USSR and S.E. Alaska may also contain small amounts of primary platinum.

TABLE 3-5. WORLD MINE PRODUCTION AND RESERVE BASE OF PLATINUM.

	Mine Production		Reserve Base
	1980	1981	
United States	3	6	16,000
Canada	405	350	9.000
South Africa, Republic of	3,100	3,100	970,000
Other Market Economy Countries	72	74	NA
U.S.S.R.	3,250	3,250	200,000
World Total	6,830	6,780	1,195,000

SOURCE: USBM, 1982b, p. 115.

Platinum placer ores, particularly in Alaska, have been produced in the past, and include the currently operating Good News Bay deposits; they have considerable potential for future mining, if exploration and discovery are encouraged by the world market. Some of these potential Alaskan placers may have very low platinum content, but mining for other ores, such as gold, may allow the recovery of some byproduct platinum as well. Under such conditions, perhaps as much as 10 to 25 million troy ounces of platinum may be eventually recoverable from the Alaskan placer deposits (see Table 2-9). The rather high cost of placer mining in Alaska, however, will act as a constraint to the broad and intensive exploitation of the Alaskan platinum placer deposits.

Except for the Stillwater and Duluth Complexes, major new platinum resource discoveries in the lower 48 of the United States appear highly unlikely because of the scarcity of layered ultramafic bodies in the crust. The exploitation of these known platinum zones would require bulk underground mining techniques or surface mining open cuts, where relatively low cost bulk extraction could be applied. It is unlikely that major new placer ores, formed from the primary deposits in Montana and Minnesota, may eventually be discovered.

The world and U.S. platinum resource estimate is only imperfectly known. Most of these estimates are based on extensions of known occurrences of platinum. The assumptions on availability are based on forecasted long-term economic conditions for the extraction of platinum. Although most geologists consider the 3.0 billion troy ounce platinum resource estimate as conservative, the estimate for ultimately recoverable platinum in the next 25-50 years may be as little as 500 million troy ounces or as high as 2.0 billion troy ounces given the uncertainty in the growth-rate of demand. It is only within an order of magnitude that we can establish the platinum resources currently identifiable.

It is also unlikely that further exploration in continental terrains will uncover large new primary resources anywhere in the world of platinum and platinum-group metals, with the possible exception of Antarctica and central Asia. Based on our understanding of the crust and the crustal association of platinum and certain rock types, it seems unlikely that many more mafic or ultramafic layered intrusions of great size will be discovered. Regardless of the origin of these bodies, i.e. meteorite impact or partial melting of large segments of the mantle due to normal but rare forces, the number yet to be discovered by geologic or geophysical tools will probably be quite small and limited to the areas of the world where even geological reconnaissance is unknown.

Chapter 3

BIBLIOGRAPHY

Dames & Moore Minerals and Resources Economics Group, 1980, Cobalt, Chromium, and Nickel Supply, Demand, and Strategic Importance. San Francisco: Dames & Moore.

Charles River Associates, 1982, Survey of the Cobalt Supply and Demand Picture (by B.J. Reddy). Paper presented at National Materials Advisory Board Committee Meeting on Technological Alternatives to Cobalt Conservation. Boston: Charles River Associates.

U.S. Bureau of Mines, 1982a, Minerals Availability System (MAS) File. Denver: U.S. Department of the Interior.

U.S. Bureau of Mines, 1982b, Mineral Commodity Summaries, Washington, D.C.: U.S. Bureau of Mines.

Chapter IV

PRODUCTION HISTORY AND REAL PRICE OF METALS

The recent production history of cobalt, chromium, nickel, manganese and platinum is reviewed in this chapter. Most of the information is derived from the U.S. Bureau of Mines "Mineral Commodity Summaries" series as well as other recently published data.

COBALT

Recent world mine production of cobalt has remained essentially constant, varying from a high of 25,000 metric tons to a low of 17,000 metric tons (Table 4-1). Most has come from Zaire and Zambia, which accounted for 60% to 70% in the period since 1975. The Philippines, where the capacity to produce cobalt has been increasing over the last six or seven years, has reached a high of 5% of the world mined total in 1980. Australian production capability since the mid-1970's has grown to about 7% of the world total. In most other producing countries, principally Finland, Canada, and Morocco, mine production of cobalt has remained stable since the mid 70's, and these countries equally divide the remainder of world production.

Although mine capacity for cobalt has remained rather static for the last five to eight years, the anticipation of modest economic recovery in the mid-1980's has generated an interest by traditional producers in expanding their current cobalt capabilities. Zaire's potential for expansion is enormous and the major mining interests in Zaire are planning to add 6,000 metric tons of cobalt capacity to the 16,000 tons that already exist there (CRA, 1982).

Zambia is the second most important source of cobalt in the western marketing framework and its production is controlled by two European-based

TABLE 4-1

MINE PRODUCTION OF COBALT IN THE NONCOMMUNIST WORLD
(Metric Tons)

	Australia	Botswana	Canada	Finland	Morocco	New Caledonia	Philippines	Zaire	Zambia	Zimbabwe	Total
1975	NA	81	1,354	1,402	1,961	NA	117	13,638	1,843	--	20,396
1976	544	132	1,356	1,278	934	82	492	10,686	2,175	--	17,679
1977	1,000	163	1,485	1,227	1,015	109	1,084	10,215	1,704	--	18,002
1978	1,360	261	1,234	1,296	1,134	154	1 191	13,125	2,063	18	21,836
1979	1,540	294	1,381	1,162	961	209	1,239	14,100	3,176	190	24,252
1980	1,600	290	1,603	1,270	1,000	180	1,270	14,500	3,304	110	25,132

Notes: Data reported for Australia and New Caledonia in 1975 are inconsistent with data for later years, so are not presented here.
Data for Zaire and Zambia are set equal to metal production data.
Data for South Africa are not available.

SOURCE: Charles River Associates, 1982, p. 10. Based on data from U.S. Bureau of Mines. Minerals Yearbook (Annual) and Mineral Industry Surveys. Washington, D.C.: U.S. Government Printing Office.

companies. In spring 1982, these companies began to add approximately 5,000 metric tons of cobalt per annum in refinery capacity (CRA, 1982). Anticipated expansion of Canadian cobalt mining and refining capacity is expected to only slightly increase current levels, for the industry has already undergone considerable expansion over the last two years, particularly at the Sherrit-Gordon Mines. Significant cobalt mining production expansion are not likely in any of the remaining producer countries in the near future; however, Dames and Moore (1980) suggest that Cuba, Uganda and the United States might be significant producers in 1985 (Table 4-2). However, our analysis (Chapters VIII and IX) does not support Dames and Moore's conclusion regarding the United States. It is more appropriate to regard Dames and Moore's projections for 1985 as longer term production potentials, within which United States possible future production can be given a context.

The cobalt ores of New Caledonia are associated with nickel laterites, and the cobalt is usually not recovered in current processing technologies. Most of the current Moroccan cobalt production is ear-marked for shipment to France and there are no plans for the immediate expansion of the approximately 1000 metric tons per year capacity. However, should new deposits be discovered in the Moroccan copper belt, both mining and processing expansion could be undertaken within the next 8 to 12 years.

The historical value of cobalt over the last two decades is presented in Table 4-3. The price of cobalt, based on constant 1978 dollars, moved upwards very slowly in the period 1960 to 1977, increasing approximately 50%. In 1978 the price of cobalt doubled and then redoubled in 1979, as international cobalt availability became limited. The product price jumped from $6.85 a pound in early 1978 to $25.00 a pound in February of 1979. At one time in

TABLE 4-2

PROJECTED PRODUCTION OF EXTRACTABLE COBALT
(tons cobalt metal)

Countries		Actual 1977	1980	1985
Australia	C	3,800	3,800	3,800
	E	—	3,040	3,040
Botswana	C	193	300	300
	E	—	270	270
Canada	C	1,697	1,650	2,700
	E	—	1,320	2,160
Cuba	C	1,763	1,600	4,500
	E	—	960+	2,700+
Finland	C	1,400	1,400	1,500
	E	—	1,120	1,200
Indonesia	C	—	475	475
	E	—	427	427
Morocco	C	1,003	2,000	2,000
	E	—	1,500	1,500
New Calendonia	C	2,000	2,000	2,000
	E	—	630	630
Philippines	C	700	1,700	1,700
	E	—	1,360	1,360
Republic Of South Africa	C	280	280	500
	E	—	250	450
Uganda	C	—	—	1,680
	E	—	—	1,344
United States	C	—	—	4,801
	E	—	—	2,822
USSR	C	2,094	2,000	2,000
	E	—	1,200	1,200
Zaire	C	14,998	12,241	18,166
	E	—	12,241	18,166
Zambia	C	2,700	3,450	4,526
	E	—	3,450	4,526
TOTAL:				
CONTAINED		32,628	32,896	50,648
EXTRACTABLE		—	27,768	41,795

Source: Dames & Moore, 1980, p. 7. Based on data from USBM Mineral Commodity Summaries, 1979, p. 39, and 1980, p. 39; Kruszona, et al., 1979, p. 60.

C = Contained
E = Extractable

TABLE 4-3

TIME, PRODUCER PRICE RELATIONSHIP FOR COBALT, 1957-1980

	Average annual price, dollars per pound	
Year	Actual prices	Constant dollars 1978 Base
1957	2.03	4.76
1960	1.54	3.41
1965	1.62	3.32
1970	2.20	3.66
1971	2.20	3.48
1972	2.45	3.73
1973	3.00	4.31
1974	3.46	4.54
1975	3.98	4.76
1976	4.44	5.05
1977	5.58	5.99
1978	11.53	11.53
1979	25.00	22.52
1980 (August)	25.00	21.64

Source: Dames and Moore, 1980, p. 15. Based on data from USBM, 1980.

that price escalation, however, cobalt was selling for over $40.00 per pound. The price has since fallen with the worldwide economic recession, and cobalt contracts are selling for about $12.00-$12.50 a pound while the spot market is considerably less.

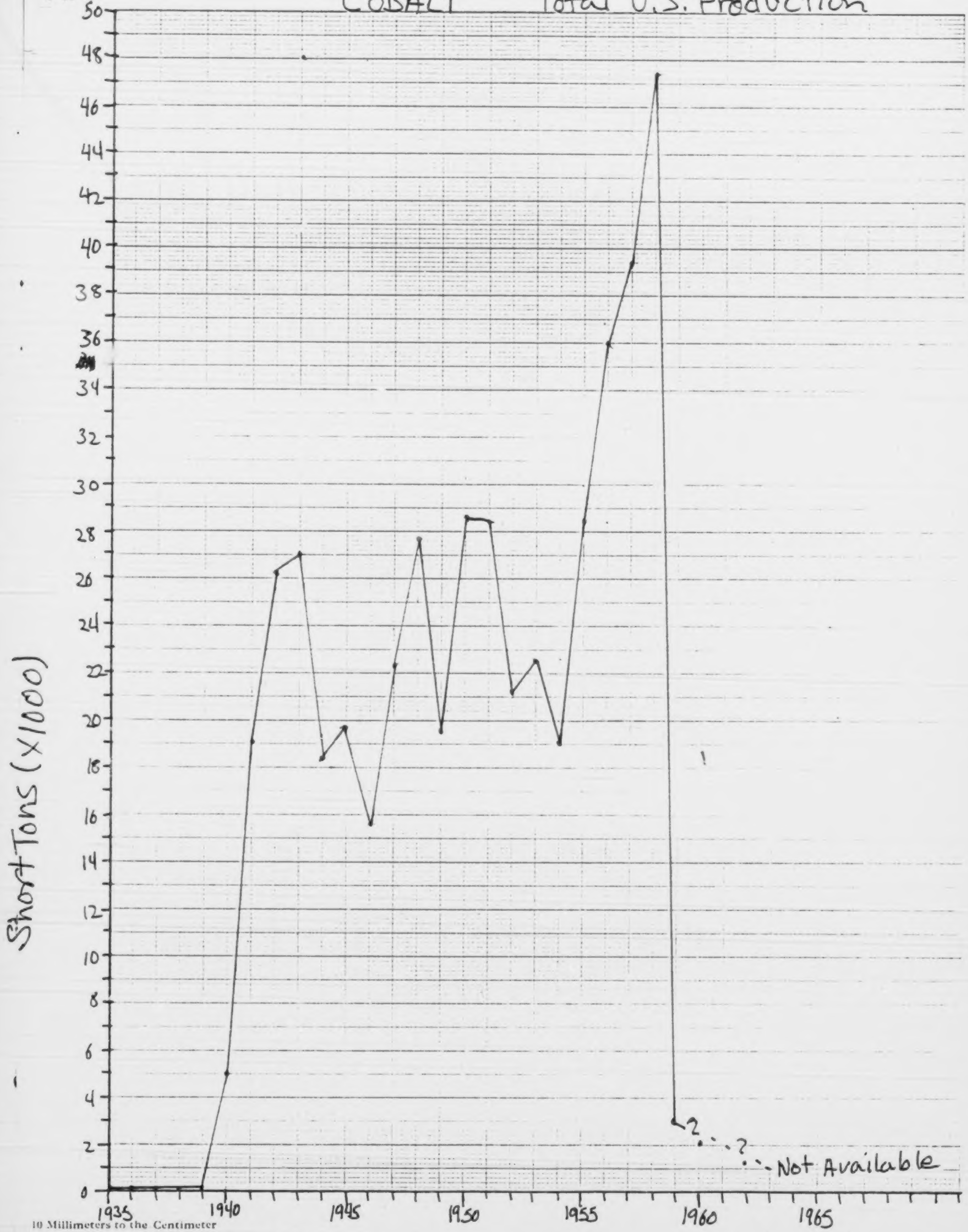
It appears now quite likely that supply will exceed demand for several years and continue the pressure toward price cutting. Such conditions are not amenable to the introduction of new cobalt capacity in either mining or refining. It is probable, therefore, that existing and planned capacity increases in cobalt production by the world's producers during the next 10 to 30 years will remain sufficient to meet most demand. Cobalt production in the United States has most recently been from the old Missouri lead belt, the Blackbird and Kellogg mines in Idaho, and the Cornwall mine in Pennsylvania. Specific production data, by year, is generally not available from the Bureau of Mines; however, Table 4-4 and Figure 4-1 summarize what available data there are. Historical data specific to the Blackbird and Madison Mines are included in Chapter V.

NICKEL

World production of nickel has declined from a 1975 high of nearly 900 thousand tons of nickel metal to a 1981 production of a little more than 700 thousand tons (Table 4-5). In the last 10 years, nickel production has been centered in Canada, the USSR, New Caledonia, and Australia, with smaller amounts coming from a variety of laterite ores and sulfides of layered strataform intrusives around the world. In 1981, the only important nickel production in the United States was from Hanna mining's Nickel Mt. Mine in Riddle, Oregon, where the richest of the nickeliferous laterites in the Coast

FIG. 4-1

COBALT Total U.S. Production



10 Millimeters to the Centimeter

Table 4-4. Breakdown of Cobalt Production by State and Region (short tons)

Year	State	Produced	Region
1935	?	23	?
1936	?	6	?
1937	?	24	?
1938	?	16	?
1939	?	27	?
1940	Idaho	93	Sullivan Mining Co., Kellog, ID
	Other	4955	?
1941	Pennsylvania	?>>	Bethlehem Steel, Cornwall, PA
	Vermont	86	Burlington
	Arizona	112	Goodsprings, Fort Thomas
	Idaho	66	Kellogg
1942	Pennsylvania	?>>	?
	Vermont	30	Burlington
	Idaho	64	Kellog
1943	Pennsylvania	?>>	?
	Vermont	23	Burlington
	Idaho	77	Kellogg
1944	Pennsylvania	>>?	?
	Vermont	?	?
	Idaho	65	Kellogg
1945	Pennsylvania	?	Cornwall
	Missouri	?	Madison County
	Idaho	?	Kellog
1946	Pennsylvania	?	Cornwall
	Missouri	?	Madison County
	Vermont	?	Burlington
1947	?	?	?
1948	Pennsylvania	?	Cornwall
1949	Pennsylvania	?	Cornwall
1950	Pennsylvania	?	Cornwall
1951	Pennsylvania	?	Cornwall
	Idaho	?	Kellog
	Missouri	?	Madison County

Table 4-4 (cont'd.)

Year	State	Produced	Region
1952	Pennsylvania	?	Cornwall
	Idaho	97	Kellog; Calera Mining Co., Lemhi County
1953	Pennsylvania	?	Cornwall
	Idaho	76	Kellog
1954	Pennsylvania	?	Cornwall
	Idaho	72	Kellog; Calera Mining Co., Lemhi County
1955	Pennsylvania	?	Cornwall
	Idaho	?	Lemhi County; Kellog
1956	Pennsylvania	?	Cornwall
	Idaho	?	Lemhi County; Kellog
	Missouri	?	Fredrickstown
1957	Pennsylvania	?	Cornwall
	Idaho	?	Lemhi County; Kellog
	Missouri	?	Fredrickstown
1958	Pennsylvania	?	Cornwall
	Idaho	?	Lemhi County; Kellog, Cobalt
	Missouri	?	Fredrickstown
1959	Pennsylvania	?	Cornwall
	Idaho	?	Lemhi County; Kellog, Cobalt
	Missouri	?	Fredrickstown

1960 - 1971 Not Available

1972 - 1980 Not Produced

Table 4-4. Breakdown of Cobalt Production by State and Region (short tons)

Year	State	Produced	Region
1935	?	23	?
1936	?	6	?
1937	?	24	?
1938	?	16	?
1939	?	27	?
1940	Idaho	93	Sullivan Mining Co., Kellog, ID
	Other	4955	?
1941	Pennsylvania	?>>	Bethlehem Steel, Cornwall, PA
	Vermont	86	Burlington
	Arizona	112	Goodsprings, Fort Thomas
	Idaho	66	Kellogg
1942	Pennsylvania	?>>	?
	Vermont	30	Burlington
	Idaho	64	Kellog
1943	Pennsylvania	?>>	?
	Vermont	23	Burlington
	Idaho	77	Kellogg
1944	Pennsylvania	>>?	?
	Vermont	?	?
	Idaho	65	Kellogg
1945	Pennsylvania	?	Cornwall
	Missouri	?	Madison County
	Idaho	?	Kellog
1946	Pennsylvania	?	Cornwall
	Missouri	?	Madison County
	Vermont	?	Burlington
1947	?	?	?
1948	Pennsylvania	?	Cornwall
1949	Pennsylvania	?	Cornwall
1950	Pennsylvania	?	Cornwall
1951	Pennsylvania	?	Cornwall
	Idaho	?	Kellog
	Missouri	?	Madison County

1952	Pennsylvania	?	Cornwall
	Idaho	97	Kellog; Calera Mining Co., Lemhi County
1953	Pennsylvania	?	Cornwall
	Idaho	76	Kellog
1954	Pennsylvania	?	Cornwall
	Idaho	72	Kellog; Calera Mining Co., Lemhi County
1955	Pennsylvania	?	Cornwall
	Idaho	?	Lemhi County; Kellog
1956	Pennsylvania	?	Cornwall
	Idaho	?	Lemhi County; Kellog
	Missouri	?	Fredrickstown
1957	Pennsylvania	?	Cornwall
	Idaho	?	Lemhi County; Kellog
	Missouri	?	Fredrickstown
1958	Pennsylvania	?	Cornwall
	Idaho	?	Lemhi County; Kellog, Cobalt
	Missouri	?	Fredrickstown
1959	Pennsylvania	?	Cornwall
	Idaho	?	Lemhi County; Kellog, Cobalt
	Missouri	?	Fredrickstown

1960 - 1971 Not Available

1972 - 1980 Not Produced

TABLE 4-5

WORLD NICKEL PRODUCTION BY COUNTRY, 1973-1979
(tons nickel metal)

Country	1973	1974	1975	1976	1977	1978	1979^e Quantity	1979^e Percent
United States	18,272	16,618	16,987	16,469	14,347	14,400	14,000	1.9
Canada	268,908	299,980	269,826	289,348	259,400	140,450	130,000	17.4
New Caledonia	109,005	142,800	146,767	117,506	120,200	72,840	85,000	11.4
Other Market Economies	142,629	188,344	247,249	256,014	255,800	293,500	308,000	41.3
Cuba^e	35,000	35,000	40,300	40,800	40,000	40,000	41,000	5.5
Other Central Economies^e	152,200	143,300	179,200	166,200	161,000	177,200	168,000	22.5
TOTAL	726,014	826,042	900,329	886,337	850,747	738,390	746,000	100.0

Source: Dames & Moore, 1980, p. 59. Based on data from USBM, Mineral Commodity Summaries, 1975-1980.

e = estimate

Range was being mined. In April of 1982 this mine closed, for reasons discussed in Chapter V.

Dames and Moore (1980) (table 4-6) has estimated future world production of nickel. Although we do not agree with their analysis for expected production in the United States in 1985, their overall estimate provides insight into global trends in expanding nickel production. Canadian nickel mine production will probably remain stable over the next 10 years and no expansion of nickel capacity is planned. The same is true for the Australian nickel ores, which are derived both from nickel laterites and massive sulfide deposits. New nickel production capacity is planned from the nickeliferous laterites of New Caledonia and Indonesia, currently scheduled to be on-line in the early 1990's. The same is true for the planned increase in nickel production in Brazil and the Cuban nickel mines at Nicaro and Moa Bay; however, the new Cuban nickel capacity may go on-line as early as 1986 or 1987, and the expansion would almost double the current Cuban mine capacity. The Soviet Union and the People's Republic of China are both known to have extensive deposits of nickeliferous ores, both of the stratiform variety and nickeliferous laterites. The deposits in the stratiform rock complex at Noril'sk in the Soviet Union are scheduled for expansion during the decade of the 80's and new mine capacity may be introduced in the People's Republic, depending upon the international market demand and home requirements. Nickel mine production capacity in the Republic of South Africa and in Zimbabwe will probably expand very slowly, if at all, and Finland's nickel production will probably remain near constant over the next decade (Dames and Moore, 1980).

Ore grades of these world nickel reserves and resources vary drastically from country to country and deposit to deposit, depending on ore type and

Fig. 4-2

TOTAL UNITED STATES NICKEL PRODUCTION (NATIONAL AND STATE)

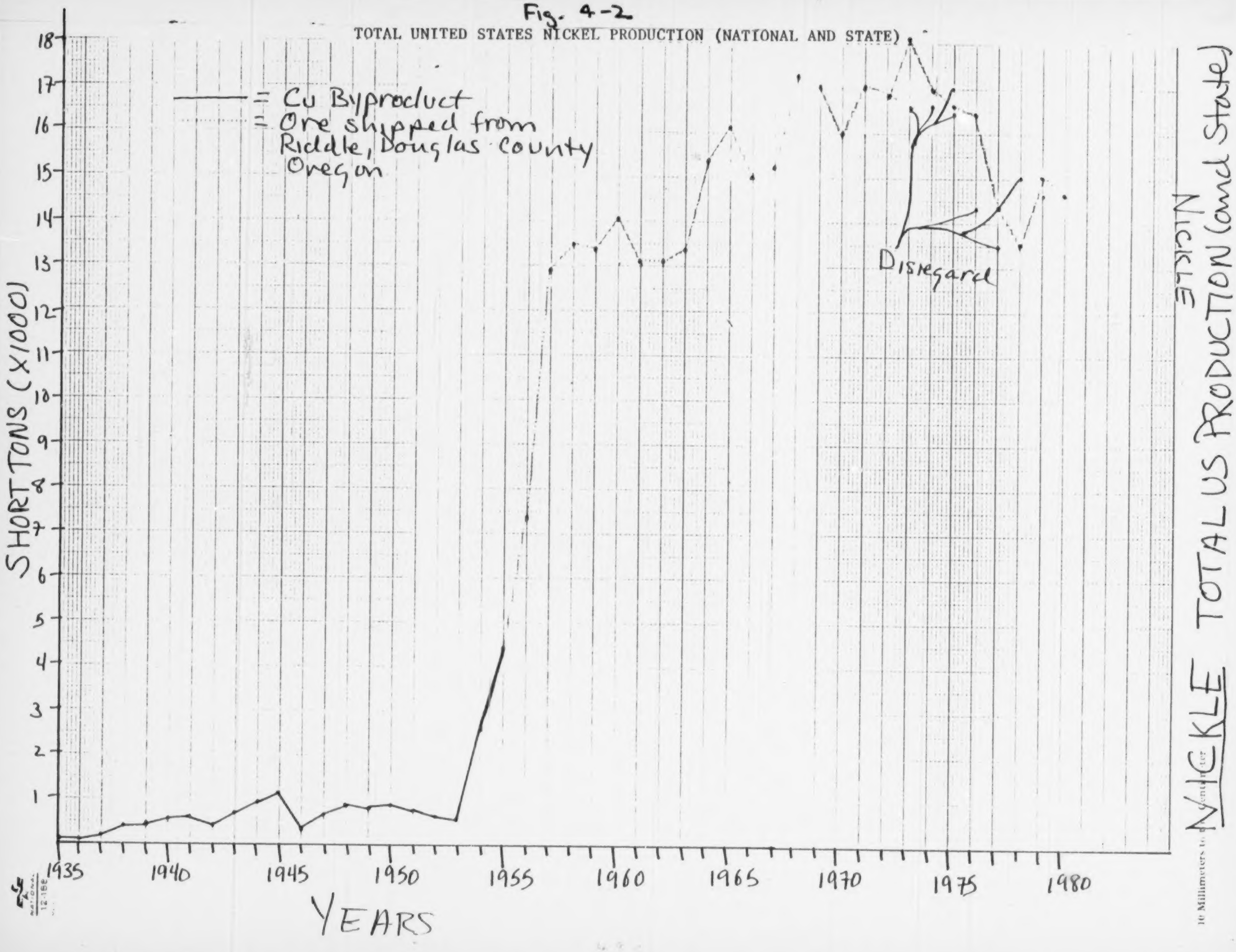


TABLE 4-6

PROJECTED MINE NICKEL PRODUCTION CAPACITY
(tons nickel)

Country	1980	1985
United States	14,000	27,700
Albania	7,000	7,000
Australia	109,100	109,000
Botswana	20,000	20,000
Brazil	3,300	5,500
Canada	275,000	275,000
China	NA	NA
Colombia	0	24,000
Cuba	42,500	52,500
Dominican Republic	32,250	32,250
Finland	9,500	9,500
Greece	17,000	17,000
Guatemala	0	14,000
Indonesia	36,300	36,300
New Caledonia	150,000	150,000
Philippines	47,000	47,000
Republic of South Africa	24,000	24,000
U.S.S.R.	185,000	210,000
Yugoslavia	0	17,600
Zimbabwe	13,000	13,000
TOTAL	984,950	1,091,350

Source: Dames & Moore, 1980, p. 60.

NA = Not Available

whether the nickel is in a primary or secondary deposit. Mine grades as low as 0.1% nickel to as high as 4.0% or 5.0% nickel will be mined in a variety of geologic environments over the next 30 years.

The price of nickel has changed little in the past 25 years (Table 4-7). In constant 1975 dollars, the price of nickel has only doubled in the period 1955 to 1982. Most recently the actual price of nickel was about $3.50 a pound, or about $3.00 per pound in 1975 constant dollars. The stable price history and large and diverse potential world supply suggests that a rapid nickel price run up will not occur in the decade of the 80's. There is a significant overcapacity in the world with respect to nickel mine production and refining, and consumer response to economic recovery is anticipated to be steady but slow.

CHROMITE

Over the last 10 years world mine production of chromite has increased by approximately 20% to 25%. A significant part of that increase has been in expansion at South African chromite mines, along with lesser expansion in Turkey and a few other market economy countries. During the decade of the 70's, chromite production from central economy countries has declined dramatically, while production from Rhodesia, Zimbabwe and the Philippines have remained essentially constant (Table 4-9). World production has dropped somewhat in the period 1979-1981. Production capacity is dominated today by the South African Bushveld chromite ores, amounting to 35% of the world total.

The United States, one of the major buyers of world chromite, has diversified its supply over the last 10 years, primarily by reducing dramatically the Soviet Union imports and increasing those from South Africa

TABLE 4-7

TIME, PRODUCER PRICE RELATIONSHIP FOR NICKEL, 1955-1980

	Average Annual Price, Dollars per Pound	
Year	Actual Price	Constant dollars, 1975 base
1955	0.64	1.35
1960	0.74	1.37
1965	0.78	1.33
1970	1.28	1.78
1971	1.33	1.76
1972	1.40	1.78
1973	1.53	1.84
1974	1.74	1.90
1975	2.03	2.03
1976	2.20	2.09
1977	2.30	2.12
1978	2.08	1.83
1979	2.49	2.20
1980 (August)	3.00	2.61

Source: Dames & Moore, 1980, p. 72. Based on data from USBM Mineral Commodity Profile, Nickel-1977; USBM Minerals and Materials, February 1980.

Table 4-8. Breakdown of Nickel Production by State and Region (short tons)

Year	State	Shipped	Region
1935			Cu byproduct in various states
1936			" " " " "
1937			" " " " "
1938			" " " " "
1939			" " " " "
1940			" " " " "
1941			" " " " "
1942			" " " " "
1943			" " " " "
1944			" " " " "
1945			" " " " "
1946			" " " " "
1947			" " " " "
1948			" " " " "
1949			" " " " "
1950			" " " " "
1951			" " " " "
1952			" " " " "
1953			" " " " "
1954			" " " " "
1955	Oregon	4111	Riddle, Douglas County*
1956	Oregon	7392	" " " "
1957	Oregon	12900	" " " "

*All ores produced at Riddle, Oregon in Douglas County by Hanna Mining Company.

Table 4-8 (cont'd.)

Year	State	Shipped	Region
1958	Oregon	13490	Riddle, Douglas County
1959	Oregon	13374	" " " "
1960	Oregon	14079	" " " "
1961	Oregon	13133	" " " "
1962	Oregon	13110	" " " "
1963	Oregon	13394	" " " "
1964	Oregon	15420	" " " "
1965	Oregon	16188	" " " "
1966	Oregon	15036	" " " "
1967	Oregon	15287	" " " "
1968	Oregon	17294	" " " "
1969	Oregon	17056	" " " "
1970	Oregon	15933	" " " "
1971	Oregon	17036	" " " "
1972	Oregon	16864	" " " "
1973	Oregon	18272	" " " "
1974	Oregon	16618	" " " "
1975	Oregon	16987	" " " "
1976	Oregon	16469	" " " "
1977	Oregon	14347	" " " "
1978	Oregon	13509	" " " "
1979	Oregon	15065	" " " "
1980	Oregon	14653	" " " "

TABLE 4-9

WORLD MINE PRODUCTION OF CHROMITE BY COUNTRY, 1974-1979
(000 tons gross weight of ore)

Country	1974	1975	1976	1977	1978	1979^e	
						Quantity	Percent
United States	0	0	W	0	0	0	0
Philippines	584	573	471	592	587	600	5.7
Rhodesia^e	650	650	670	660	660	660	6.2
South Africa	2,069	2,285	2,656	3,656	3,466	3,500	33.0
Turkey	550	739	816	700	750	750	7.1
Other Market Economy Countries	1,176	1,360	1,622	1,776	2,551	2,500	23.6
Central Economy Countries	2,902	3,134	3,257	3,420	2,546	2,600	24.5
TOTAL	7,931	8,741	9,492	10,804	10,560	10,610	100.0

Source: Dames & Moore, 1980, p. 29. Based on data from USBM, Mineral Commodity Summaries, 1975-1980.

Note: Column items may not add to totals because of rounding.

W = Withheld to avoid disclosing company confidential data.

e = estimate.

and other free market economies (Table 4-10). At the same time U.S. imports of ferrochromium have increased significantly from South Africa, remained constant from Zimbabwe and Japan, and generally declined from other sources (Table 4-11). There appears to be little current incentive to dramatically increase the availability of chromite from traditional or new sources. Reserves of the major producing countries appear to be substantial, particularly those from Zimbabwe and the Republic of South Africa. With the lower current demand for chromium and large mine overcapacity, it is unlikely that significant expansion will take place, even with a modest recovery in the 80's, anywhere but in South Africa and Zimbabwe.

With the world price of chromite at a near decade low, it can now be delivered to the United States coast for approximately $100.00 per ton from African or Turkish sources. Over the last 25 years the constant dollar value of chromite has approximately doubled, reaching a peak in 1977 and 1978 (Table 4-12). Since that time, the constant 1978 dollar value of chromite has declined by nearly 50%.

United States production of chromite most recently took place in the Stillwater Complex of Montana, as is discussed in Chapter V. This mining continued untilk 1961. Prior to 1958, scattered small disseminated/podiform deposits in Alaska, California, Oregon, and Washington were mined. Production in any of these states never was more than 45% of the production in the Stillwater Complex in the same year (table 4-13, figure 4-3).

MANGANESE

In the last two decades the United States, the USSR, Japan and South Africa have been the major consumers of manganese ore (Table 4-14), and the

TABLE 4-12

TIME, VALUE RELATIONSHIP FOR CHROMIUM

Year	Value, dollars per ton[a] Actual	Constant 1978 dollars
1954	82	209
1957	83	194
1960	62	137
1963	49	104
1965	54	111
1968	53	98
1970	72	120
1971	79	125
1972	81	123
1973	74	106
1974	87	114
1975	159	190
1976	192	218
1977	190	204
1978	184	184
1979	130	120
1980 (August)	119	103

Source: Dames & Moore, 1980, p. 43. Based on data from USBM, Mineral Commodity Profile, Cobalt - 1980.

[a] Average custom value per ton of chromium in imported chromite.

Table 4-13. Breakdown of Chromium Production by State and County
(long tons: 1935-41)
(short tons: 1942-80)

Year	State	Shipped	Region (Counties)
1935	California	440	Fresno, Napa, Placer, San Luis Obispo, Santa Barbara, Shasta, Siskiyou
1936	California	269	Del Norte, Fresno, Eldorado, Napa, San Luis Obispo, Santa Barbara
1937	California	2033	Del Norte, Eldorado, Nevada, Napa, San Luis Obispo, Shasta, Santa Barbara
	Oregon	288	Josephine
1938	California	812	Eldorado, Napa, Placer
1939	California	3614	Plumas, Calaveras, Placer, Del Norte, Siskiyou, Tuolomne, Eldorado
	Oregon	<<	Grant
1940	California	2662	Eldorado, Fresno, Del Norte, Placer, Plumas, Siskiyou, Shasta, Grant
	Oregon	<<	Grant
1941	California	1273	Del Norte, Fresno, Eldorado, Placer, Humbolt
	Oregon	<<	Curry, Jackson, Josephine
1942	California	44873	Glenn, Eldorado, San Luis Obispo, Del Norte, Tehama, Placer, Siskiyou, Humbolt
	Oregon	2683	Grant, Curry, Josephine
	Montana	>1/2 U.S. tot.	Stillwater County
	Idaho	?	Idaho
1943	California	62495	Glenn, San Luis Obispo
	Alaska	5569	Kenai Penninsula
	Montana	75691	Stillwater
	Oregon	7818	Curry, Coos
1944	California	34715	Del Norte, Fresno, Eldorado, Glenn, Humbolt, Placer, Shasta, San Luis Obispo, Siskiyou, Tehama
	Alaska	1845	Kenai Penninsula
	Montana	1251	Madison
	Oregon	7818	Curry, Grant, Jackson, Josephine
1945	California	9607	Del Norte, Placer, Eldorado, Shasta, San Luis Obispo, Siskiyou, Tehama, Butte, Calaveras, Fresno, Humbolt, San Benito

Table 4-13 (cont'd.)

Year	State	Shipped	Region (Counties)
1946	California	4107	Del Norte, San Luis Obispo
1947	California	948	Del Norte, Butte
1948	California	274	Del Norte
	Oregon	3345	Josephine
1949	California	443	Butte, Tehama, Del Norte
1950	California	404	?
1951	California	6302	Del Norte, Butte, Siskiyou, Fresno, Shasta
	Oregon	754	Grant, Curry, Jackson, Josephine
1952	California	14713	Siskiyou, Del Norte, San Luis Obispo
	Oregon	6591	Josephine
1953	California	26512	San Luis Obispo, Siskiyou, Fresno
	Montana	26089	Stillwater
	Oregon	6216	Curry, Grant, Josephine
1954	Alaska	7082	Kenai Penninsula
	California	30661	San Luis Obispo, Butte, Siskiyou, Fresno
	Montana	123096	Stillwater
	Oregon	6655	Josephine, Grant, Curry
1955	Alaska	7082	Kenai Penninsula
	California	22105	Del Norte, Siskiyou, San Luis Obispo, Butte, Fresno
	Montana	118703	Stillwater
	Oregon	5341	Josephine, Curry, Grant
	Washington	22	Okangan
1956	Alaska	7193	Kenai Penninsual
	California	27082	Butte, Fresno, the Sierra Nevada Coast, Klamath Mts.
	Montana	118720	Stillwater
	Oregon	54577*	*includes 45710 ST from Government stockpile--Josephine, Curry, Grant
	Washington	30	Okangan
1957	Alaska	4207	Kenai Penninsula
	California	34901	Del Norte, Butte, Siskiyou, Fresno, San Luis Obispo
	Montana	119149	Stillwater
	Oregon	7900	Josephine, Grant

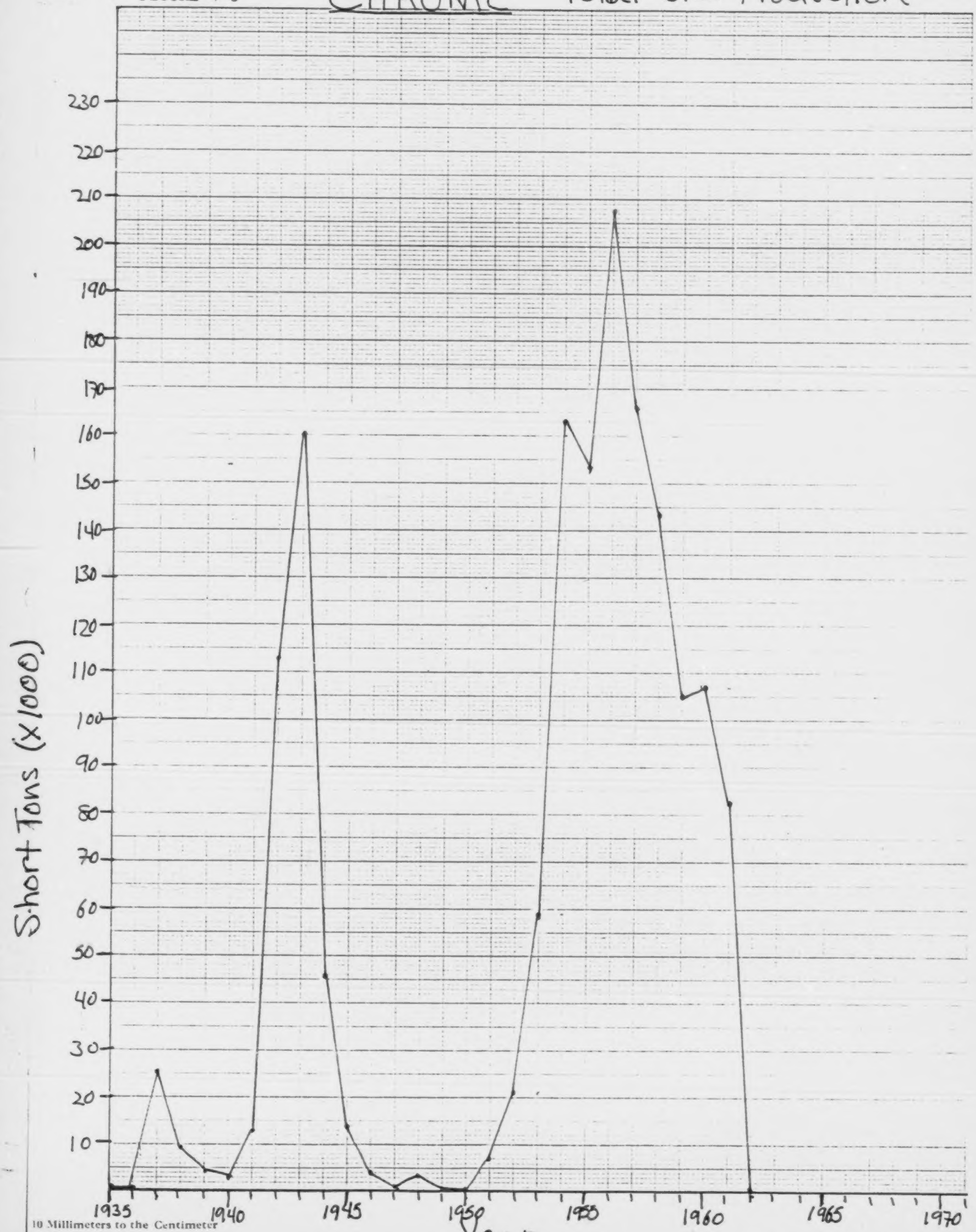
Year	State	Shipped	Region (Counties)
1958	California	20588	San Luis Obispo, Fresno
	Montana	119057	Stillwater
	Oregon	4133	Douglas, Grant, Curry, Josephine, Jackson
	Washington	17	Skagit
1959	Montana	105000	Stillwater
1960	Montana	107000	Stillwater
1961	Montana	82260	Stillwater

1962 - 1980 No Production

NATIONAL
12-188

FIGURE 4-3

CHROME Total U.S. Production



10 Millimeters to the Centimeter

52F

United States, the USSR, and Japan, the major consumers of ferromanganese (Table 4-15). Production, on the other hand, has been centered in the USSR and the Republic of South Africa, with smaller amounts coming from Brazil, Australia, Gabon and other market economies. U.S. mine production has been practically nonexistent since the expiration of susbsidies in the early 1960's.

Sedimentary manganese ores are widely dispersed in continental regimes and provide an enormous resource base upon which new mining could be developed. Worldwide projected growth rates for manganese consumption of between 2.5% to 3.0% a year (NAS, 1981) for the next 20 years would require a doubling of manganese mining capacity during this period (Table 4-16). More than likely, new manganese mines in Brazil, Gabon, and perhaps the People's Republic of China will join expanded production from known districts in Australia and the Republic of South Africa for any needed increase in supply.

Real prices of manganese, like those of chromium and nickel, have not changed dramatically in the last 20 years (Table 4-17). The abundance of new manganese sources internationally is expected to dampen and price increases induced by rising demand. Sufficient diversity of supply exists in the world so that any short-term, and possibly even long-term (20 years), interruption of manganese from unstable or politically hostile sources such as the USSR will still not significantly escalate the price of manganese. The most likely scenario for manganese is that few, if any, new mining environments will be exploited until a more specific program and policy for development of deep sea manganese nodules is accepted by the major industrial countries through the United Nations.

Future expansion of ferromanganese capacity is also difficult to project

Table 4-14. WORLD APPARENT CONSUMPTION OF MANGANESE ORE, 1960-77, BY COUNTRY[a]

(THOUSAND METRIC TONS, MANGANESE CONTENT)

YEAR	PRINCIPAL INTERNATIONAL TRADERS									OTHER COUNTRIES			WORLD TOTAL[b]
	UNITED STATES	JAPAN	FRANCE	GERMANY (F.R.)	NORWAY	AUSTRALIA	BRAZIL	SOUTH AFRICA	TOTAL	USSR	INDIA	OTHER	
1960	844	224	332	135	139	4	46	129	1913	1716	27	844	4500
1961	773	266	302	230	98	20	65	232	1988	1776	141	854	4751
1962	760	265	30[illegible]	223	124	3	104	259	2116	1904	221	768	5007
1963	790	238	339	312	105	32	167	176	2149	1987	109	824	5069
1964	851	289	347	305	159	40	234	220	2445	2141	[c]110	979	5675
1965	1251	493	378	293	229	56	147	267	3116	2295	197	941	6549
1966	1057	496	367	342	220	134	223	167	3066	2271	285	996	6558
1967	1046	650	304	269	243	150	369	232	3263	2073	313	989	6638
1968	985	705	396	370	299	168	436	173	3531	1894	230	1083	6738
1969	1038	801	436	259	282	143	517	232	3708	2026	378	1176	7288
1970	975	995	494	276	247	51	202	323	3563	1974	491	1236	7264
1971	869	1235	485	327	338	168	362	272	4056	2074	445	1298	7871
1972	1170	1145	505	301	334	192	397	259	4183	2282	333	1227	8025
1973	962	1231	645	296	376	328	372	189	4459	2431	376	1451	8717
1974	870	1430	623	333	485	327	133	525	4726	2335	324	1473	8858
1975	844	1348	503	238	483	285	291	931	4973	2470	396	1342	9181
1976	785	840	459	266	439	406	471	915	4585	2505	[d]400	1455	9025
1977	1011	[d]561	N.A.	N.A.	N.A.	N.A.	N.A.	N.A.	--	2430	N.A.	--	N.A.

SOURCE: CANADA DEPARTMENT OF ENERGY, MINES AND RESOURCES, CANADIAN MINERALS YEARBOOK; DEUTSCHES INSTITUT FUR WIRTSCHAFTSFORSCHUNG, MANGAN; JAPANESE MINISTRY OF FINANCE, JAPAN EXPORTS AND IMPORTS; ORGANIZATION FOR ECONOMIC COOPERATION AND DEVELOPMENT, FOREIGN TRADE STATISTICS, SERIES C; U.K. INSTITUTE OF GEOLOGICAL SCIENCES, STATISTICAL SUMMARY OF THE MINERAL INDUSTRY AND WORLD MINERAL STATISTICS 1970-1974 AND 1972-1976; U.S. BUREAU OF MINES, MINERALS YEARBOOK, MINERAL TRADE NOTES (VARIOUS ISSUES), AND UNPUBLISHED DATA; UNITED NATIONS STATISTICAL OFFICE, WORLD TRADE ANNUAL.

NOTE: DETAIL MAY NOT ADD TO TOTALS OWING TO ROUNDING.

N.A. - NOT AVAILABLE.

[a] INCLUDES ORES OF 10 PERCENT OR GREATER CONTENT.

[b] EXCLUDES ESTIMATES FOR THE PEOPLES' REPUBLIC OF CHINA AND ROMANIA.

[c] ESTIMATE BASED ON FERROMANGANESE PRODUCTION.

[d] PRELIMINARY.

Source: Fischman, RFF, 1981, p. 60.

53 A

Table 4-15. WORLD APPARENT CONSUMPTION OF FERROMANGANESE, 1960-77, BY COUNTRY[a]

(THOUSAND METRIC TONS, GROSS WEIGHT[b])

YEAR	PRINCIPAL INTERNATIONAL TRADERS								OTHER COUNTRIES			WORLD TOTAL[d]
	UNITED STATES	JAPAN	BELGIUM[c]	FRANCE	GERMANY (F.R.)	ITALY	UNITED KINGDOM	TOTAL	USSR	INDIA	OTHER	
1960	861	115	115	146	331	55	217	1780	592	33	183	2588
1961	864	172	114	150	327	61	173	1861	647	46	181	2735
1962	736	146	115	154	324	69	192	1736	756	99	223	2814
1963	860	152	93	145	295	82	157	1784	764	96	284	2928
1964	941	202	135	165	315	71	207	2036	856	38	521	3451
1965	1164	205	110	163	312	91	244	2289	849	93	609	3840
1966	1139	222	110	161	293	93	206	2226	815	127	552	3720
1967	1000	307	115	144	287	104	205	2162	824	107	593	3686
1968	868	339	133	137	364	142	204	2147	845	76	664	3732
1969	1081	396	112	180	337	138	246	2490	773	40	572	3879
1970	989	404	150	181	373	131	233	2541	850	86	683	4162
1971	861	517	160	207	290	135	214	2412	731	146	748	4037
1972	946	503	164	154	339	120	233	2459	730	85	710	3984
1973	1159	613	166	199	382	107	224	2930	745	114	744	4533
1974	1214	586	195	168	408	222	206	2999	831	126	968	4924
1975	839	622	130	173	332	191	180	2367	829	128	1126	4450
1976	1032	512	115	130	340	193	240	2570	701	146	950	4367
1977	807	492	N.A.	N.A.	N.A.	N.A.	N.A.	--	N.A.	N.A.	--	N.A.

SOURCE: CANADA DEPARTMENT OF ENERGY, MINES AND RESOURCES, CANADIAN MINERALS YEARBOOK; DEUTSCHES INSTITUT FUR WIRTSCHAFTSFORSCHUNG, MANGAN; JAPANESE MINISTRY OF FINANCE, JAPAN EXPORTS AND IMPORTS; ORGANIZATION FOR ECONOMIC COOPERATION AND DEVELOPMENT, FOREIGN TRADE STATISTICS, SERIES C; U.K. INSTITUTE OF GEOLOGICAL SCIENCES, STATISTICAL SUMMARY OF THE MINERAL INDUSTRY AND WORLD MINERAL STATISTICS 1970-1974 AND 1972-1976; U.S. BUREAU OF MINES, MINERALS YEARBOOK, MINERAL TRADE NOTES (VOLUME 74, NUMBER 10), AND UNPUBLISHED DATA; UNCTAD SECRETARIAT, CONSIDERATION OF INTERNATIONAL MEASURES OF MANGANESE; UNITED NATIONS STATISTICAL OFFICE, WORLD TRADE ANNUAL.

N.A. - NOT AVAILABLE.

[a] INCLUDES HIGH-CARBON, MEDIUM-CARBON, AND LOW-CARBON FERROMANGANESE.

[b] AN ESTIMATE OF ACTUAL MANGANESE CONTENT CAN BE OBTAINED BY MULTIPLYING THE FIGURES SHOWN BY 78 PERCENT.

[c] ACTUALLY, BELGIUM AND LUXEMBOURG.

[d] EXCLUDES ESTIMATES FOR THE PEOPLES REPUBLIC OF CHINA.

SOURCE: Fischman, RFF, 1981, p. 61.

Table 4-16. Projections of World Manganese Mining Capacity,[a] 1980-2000, Compared with 1975 Capacity and with Estimated and Projected Manganese Ore Consumption

(million metric tons of manganese content)

	1975[b]	1980[b]	1985	1990	1995	2000
Principal int'l manganese ore suppliers:						
South Africa	2.36	2.72	3.0	3.3	3.6	4.0
Gabon	1.12	1.29	2.3	2.3	2.3	2.3
Australia	.82	1.45	1.9	1.7	1.4	1.0
Total	4.30	5.46	7.2	7.3	7.3	7.3
Other principal producers:						
USSR	3.81	4.62	4.8	4.8	4.7	4.6
India	.73	.73	.9	1.0	1.2	1.4
Brazil	1.20	1.59	2.0	2.6	3.0	3.5
All principal producers	10.04	12.40	14.9	15.7	16.2	16.8
Percent of world capacity	88.5	90.7	90	88	85	82
World capacity	11.34	13.67	16.6	17.8	19.1	20.5
Cf. World consumption	9.00	10.30	11.6	13.1	15.0	17.1

[a] Or beneficiating capacity, if this is the constraining element.

[b] Source of 1975 estimates and 1980 projections is Gilbert L. DeHuff and Thomas S. Jones, Manganese (U.S. Bureau of Mines, Mineral Commodity Profiles, 7, Oct. 1977 and draft, May 1978).

Source: Fischman, RFF, 1981, p. 299.

Table 4-17. Trends in Manganese Prices and Costs, 1970-2000

	Historical[a]		Projected				
	1970	1975	1980	1985	1990	1995	2000
Manganese Ore ($ per long ton unit)							
Average price, metallurgical ore, delivered U.S. ports	.57	1.17	1.45	1.60	1.65	1.75	1.90
C.f, IBRD projection[b]	.60	1.26	1.53	1.98	2.41	-	-
Average cost to U.S. electro-metallurgical plants	[c].70	[c]1.38	1.70	1.85	1.95	2.05	2.20
Ferromanganese, Standard ($/L.T.)							
Aver. price delivered Pittsburgh or Chicago:							
Imported	164	339	445	475	4~~70~~	465	475
Composite, imported and domestic	173	344	450	485	475	475	480

[a] Five-year averages, centered on year shown.

[b] Unofficial staff document, May 1979. The Bank's projection is said to be for Indian ore, which is actually no longer shipped to the United States, but its series nevertheless serves as an indicator of how Bank staff view the manganese price outlook. The figure shown for 1980 is the average of the Bank's figures for 1978-79 actual and 1980-82 projections.

[c] Estimated on the basis of relationship to average value of imports in economic-census years.

Source: Fischman, RFF, 1981, p. 429.

with a high measure of assurance (Table 4-18). World ferromanganese capacity is expected to expand slightly in South Africa and the USSR over the next 10-25 years. Modest expansion can also be expected in Brazil. Total world capacity may reach one-half that of manganese ores by the end of the century. As ferromanganese expansion grows to meet steel production requirements, it seems reasonable to assume that only those countries with an expanding and vigorous steel industry will share in the growth. It also seems reasonable to conclude that future small price rises, or small, additional exploration and development efforts will bring substantial new quantities of reserves on-line, primarily in the areas where relatively high-grade manganese resources have already been identified. Any increase in manganese costs will most likely have a greater impact on supply than on consumption, for manganese is already quite cheap relative to the value of steel and probably is being used to capacity.

PLATINUM

Platinum group metals are produced in abundance from only two countries in the world, the Republic of South Africa and the USSR. Historically, mine production has been centered in these two places with an order of magnitude smaller amount produced in Canada, and two to three orders of magnitude smaller amounts produced in a variety of countries from either placer deposits or as byproduct from mining ores in basic and ultrabasic rocks. In 1981, the world production of platinum-group metals was nearly 7 million troy ounces and was essentially unchanged from the previous few years.

Because many of the placer deposits are small and are exploited to their full value, world production capacity in a stable market varies primarily as a function of the discovery of new placer mining sources. Platinum and

Table 4-18 **Projections of World Ferromanganese Capacity, 1980-2000, Compared with 1986 Capacity and with Estimated and Projected Ferromanganese Consumption**

(million metric tons)

	1976[a]	1980	1985	1990	1995	2000
Principal international suppliers:						
France	.6	.6	.6	.6	.6	.6
Norway	.4	.4	.4	.4	.4	.4
South Africa	.5[b]	.7	.8	.9	1.0	1.1
Japan	1.0	.7	.7	.8	.8	.8
Total	2.5	2.4	2.5	2.7	2.8	2.9
Other principal producers:						
United States	.6	.3	.4	.4	.4	.4
USSR	1.0[b]	1.2	1.4	1.6	1.7	1.8
Brazil	.1	.2	.4	.6	.7	.8
All principal producers	4.2	4.1	4.7	5.3	5.6	5.9
Percent of world capacity	72	72	72	72	71	70
World capacity	5.8	5.7	6.5	7.4	7.9	8.4
Cf. World consumption	4.4[c]	4.9	5.5	6.1	7.0	8.1

[a] Bundesanstalt für Geowissenschaften und Rohstoffe and Deutsches Institut für Wirtschaftsforschung, Untersuchungen über Angebot und Nachfrage mineralischer Rohstoffe, vol. VIII, Mangan (Berlin, 1977) unless otherwise noted includes silicomanganese, which is an appreciable proportion of total capacity in Japan, Norway, and Spain. Several countries adjusted to exclude amounts estimated to be devoted to silicomanganese.

[b] Estimated on the basis of output or other indicators.

[c] Partially estimated.

Source: Fischman, RFF, 1981, p. 303.

FIG. 4-4. TOTAL U.S. PRODUCTION OF MANGANESE

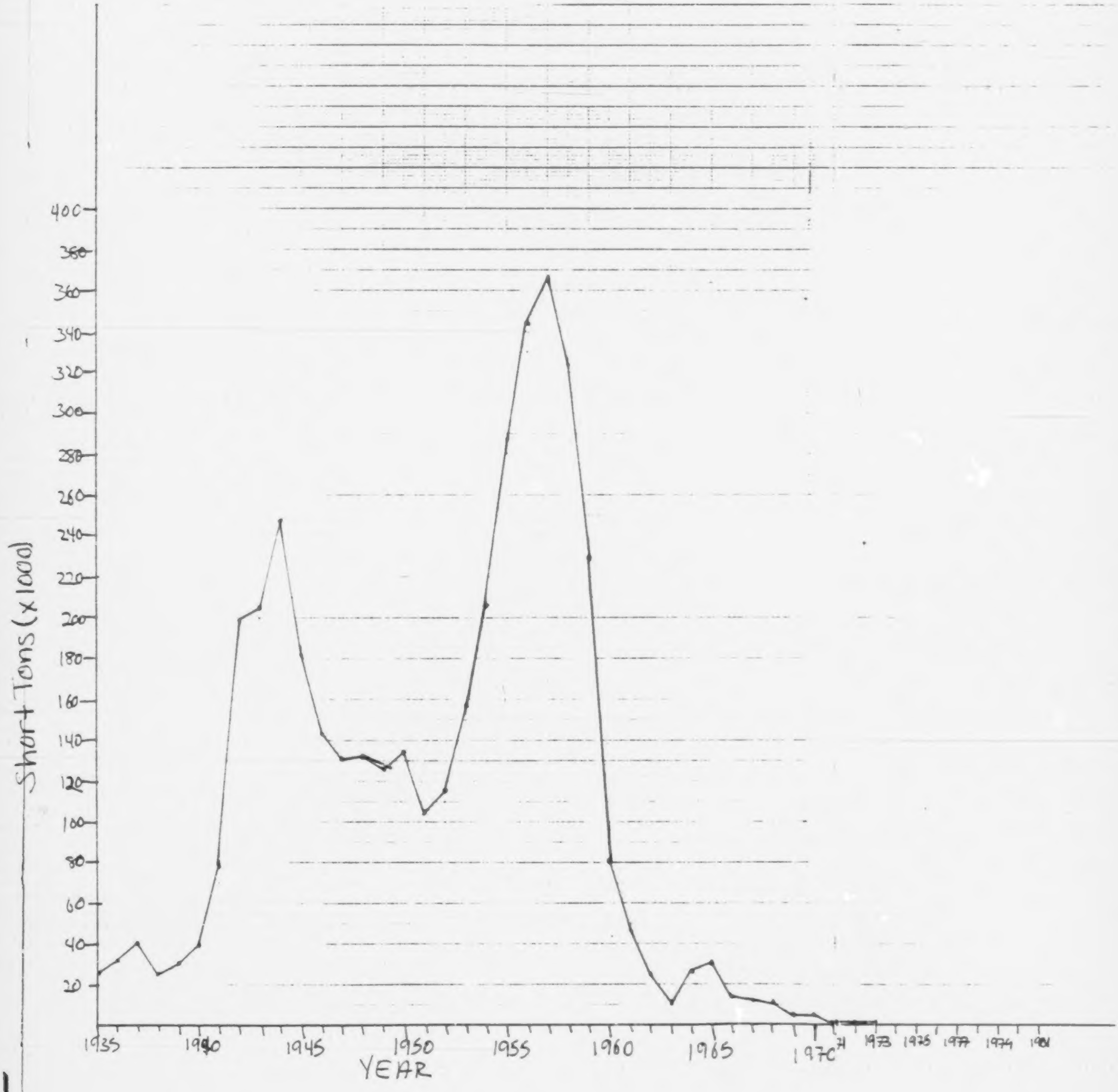


Table 4-19. Breakdown of Manganese Production by State
(long tons)

Year	State	35%	35+% Ores	Miscellaneous
			Battery	
1935	Alabama	59	—	126
	Arkansas	2809	—	—
	California	306	—	--
	Georgia	6960	—	--
	Montana	2155	7264	1404
	Tennessee	1418	—	475
	Virginia	1972	—	480
1936	Alabama	377	--	195
	Arkansas	4557	—	--
	Georgia	3821	--	—
	Montana	5154	7747	3555
	Tennessee	2679	—	860
	Utah	1635	—	--
	Virginia	196	—	1165
	West Virginia	138	—	40
1937	Alabama	31	—	258
	Arkansas	3931	—	--
	Georgia	689	—	—
	Montana	16854	6447	3443
	New Mexico	878	—	—
	Tennessee	1214	—	2361
	Texas	38	—	--
	Utah	32	—	--
	Virginia	952	—	1313
	West Virginia	1800	—	--
1938	Alabama	111	--	91
	Arkansas	2987	—	--
	Georgia	3058	--	--
	Montana	5300	4959	1677
	New Mexico	560	—	—
	Tennessee	3603	--	527
	Virginia	1314	--	928
	West Virginia	56	—	107
	Nevada	--	—	43
1939	Alabama	103	—	84
	Arkansas	5365	—	--
	Georgia	2646	--	--
	Montana	2243	7767	1129
	New Mexico	339	--	--
	North Carolina	43	--	--
	Tennessee	7306	--	529
	Utah	50	--	—
	Virginia	475	--	1186

Table 4-19 (Page 2)

			35% Ores	
Year	State	35%	Battery	Miscellaneous
	West Virgina	—	—	26
	Washington	10	—	—
	California	—	—	6
1940	Alabama	57	—	186
	Arizona	311	—	58
	Arkansas	6079	—	—
	California	158	—	—
	Colorado	224	—	—
	Georgia	3572	—	—
	Montana	8230	9271	1842
	Nevada	210	—	—
	New Mexico	45	—	—
	Tennessee	6983	00	435
	Utah	27	—	—
	Virginia	1043	—	1173
	West Virginia	219	—	—
1941	Alabama	—	—	165
	Arizona	884	20	—
	Arkansas	4478	—	—
	California	2321	976	360
	Colorado	170	—	—
	Georgia	4369	—	74
	Idaho	30	—	—
	Missouri	11	—	—
	Montana	38888	9182	649
	Nevada	2622	—	—
	North Carolina	31	—	—
	Oklahoma	36	—	—
	Oregon	76	—	—
	Tennessee	3923	—	29
	Utah	38	—	—
	Virginia	4449	—	989
	West Virginia	2195	—	5
	Washington	1418	—	—
1942	Alabama	26	—	—
	Arizona	2946	—	—
	Arkansas	4132	—	—
	California	10112	2856	—
	Georgia	4890	—	27
	Missouri	239	—	—
	Montana	120409	9521	156
	Nevada	6112	—	—
	New Mexico	1267	—	—
	Pennsylvania	22	—	—
	South Dakota	81	—	—

Table 4-19 (Page 3)

Year	State	35%	35+% Ores Battery	Miscellaneous
	Tennessee	2247	—	—
	Utah	970	—	—
	Virginia	11024	—	222
	West Virginia	2240	—	—
	Washington	10660	—	—
	Undist.	589	3033	68
1943	Arizona	5779	—	—
	Arkansas	5319	—	—
	California	20604	2625	—
	Colorado	707	—	—
	Georgia	2467	22	—
	Idaho	36	—	—
	Missouri	180	—	—
	Montana	130789	7326	—
	Nevada	10451	—	—
	New Mexico	469	—	—
	North Carolina	140	—	—
	Oklahoma	265	—	—
	Oregon	143	—	—
	South Carolina	312	—	—
	South Dakota	12	—	—
	Tennessee	2501	—	104
	Utah	91	—	—
	Virginia	7040	—	—
	Washington	7731	—	—
	Wyoming	60	00	00
	Undist.	—	2731	—
1944	Alabama	49	—	—
	Arizona	8519	—	—
	Arkansas	7109	—	—
	California	21540	—	—
	Georgia	1135	—	—
	Montana	153665	6224	—
	Nevada	21799	—	—
	New Mexico	273	—	—
	South Carolina	1400	—	84
	Tennessee	418	—	—
	Utah	30	—	—
	Virginia	20034	—	138
	Washington	5199	—	—
	Undist.	—	—	155
1945	Alabama	32	—	—
	Arizona	1093	—	—
	Arkansas	6663	—	—
	California	1668	—	—
	Georgia	1056	—	—

Table 4-19 (Page 4)

Year	State	35%	35+% Ores Battery	Miscellaneous
	Montana	143888	8042	--
	Nevada	960	—	--
	New Mexico	3334	—	--
	South Carolina	41	—	--
	Virginia	8566	—	--
	Washington	6994	—	—
	Undist.	100	—	--
1946	Arkansas	1101	—	—
	Montana	129227	8295	--
	Nevada	1064	—	--
	New Mexico	1166	—	--
	South Carolina	78	—	—
	Virginia	321	—	959
	Washington	1424	—	—
1947	Arizona	133	—	--
	Arkansas	841	—	--
	Montana	123490	6189	10
	Nevada	67	—	—
	New Mexico	858	—	--
	Tennessee	39	—	--
1948	Arizona	240	—	--
	Arkansas	212	—	—
	Montana	119339	10845	--
	Tennessee	37	—	—
	Virginia	—	—	427
1949	Arizona	223	—	--
	Arkansas	2851	—	--
	California	280	—	—
	Montana	107399	14983	--
	Tennessee	175	—	—
	Virginia	—	—	224
1950	Alabama	138	—	--
	Arizona	222	—	--
	Arkansas	1224	—	—
	California	37	—	--
	Montana	119694	11507	--
	New Mexico	1320	--	--
	Tennessee	133	—	--
	Utah	120	--	--
	Virginia	56	—	--

Table 4-19 (Page 5)

Year	State	35%	35+% Ores Battery	Miscellaneous
1951	Arizona	173	—	--
	Arkansas	3718	—	--
	Montana	291080	9482	--
	Nevada	58	270	—
	New Mexico	226	—	--
1952	Arizona	203	—	—
	Arkansas	2246	—	--
	California	2589	4492	—
	Montana	90772	9298	--
	Nevada	105	590	—
	New Mexico	2360	—	--
	Tennessee	126	—	—
	Texas	56	—	--
	Utah	95	—	—
	Virginia	1011	—	--
	Washington	436	—	--
1953	Arkansas	6123	—	--
	California	720	4693	—
	Montana	102878	10551	--
	Nevada	18368	1782	--
	Oregon	46	—	--
	Tennessee	2625	—	—
	Utah	—	55	--
	Virginia	8454	—	—
	Undist.	746	—	--
1954	Arkansas	13728	—	—
	California	393	380	58
	Montana	44735	13926	—
	Nevada	—	363	--
	Tennessee	11823	—	--
	Utah	—	25	--
	Virginia	22678	—	--
	Undist.	98019	—	--
1955	Arizona	1396	48	--
	Arkansas	13744	—	--
	California	3136	—	--
	Montana	94762	11264	--
	Nevada	101070	399	—
	New Mexico	1390	--	--
	Tennessee	15895	--	--
	Virginia	32654	--	--
	Undist.	1497	—	--

54G

Table 4-19 (Page 6)

Year	State	35%	35+% Ores Battery	Miscellaneous
1956	Arizona	42008	--	--
	Arkansas	29485	--	--
	California	6595	--	--
	Montana	77573	2979	--
	Nevada	121017	465	--
	New Mexico	22011	--	--
	Tennessee	17821	--	--
	Virginia	20231	--	--
	Undist.	4550	--	--
1957	Arizona	79505	--	--
	Arkansas	23261	--	--
	California	9009	--	--
	Colorado	175	--	--
	Montana	66191	2107	--
	Nevada	129046	--	--
	New Mexico	25456	--	--
	Tennessee	12938	--	--
	Utah	142	--	--
	Virginia	12655	--	--
	Undist.	5849	--	--
1958	Arizona	62279	--	--
	Arkansas	22221	--	--
	California	17644	--	--
	Colorado	210	--	--
	Montana	53123	--	--
	Nevada	127322	--	--
	New Mexico	24665	--	--
	Tennessee	5935	--	--
	Utah	1043	--	--
	Virginia	8128	--	--
	Undist.	538	--	--
1959	Arizona	68183	--	--
	Arkansas	17742	--	--
	California	19354	--	--
	Colorado	1218	--	--
	Georgia	1547	--	--
	Montana	15569	6011	24
	Nevada	56586	--	--
	New Mexico	27528	--	--
	Tennessee	7586	--	--
	Utah	1511	--	--
	Virginia	6232	--	--
	Washington	83	--	--
1960	Arizona	1626	--	--
	Montana	19920	9116	--
	Nevada	49076	--	--
	Tennessee	283	--	--

Table 4-19 (Page 7)

Year	State	35%	35+% Ores Battery	Miscellaneous
1961	Montana	10673	6832	10
	Nevada	28573	—	--
1962	Montana	19007	5729	22
1963	Montana	2040	3220	—
	New Mexico	5362	—	--
1964	Montana	8400	6932	—
	New Mexico	5794	—	--
	Undist.	4932	—	—
1965	Montana	17234	6387	--
	New Mexico	5637	—	—
1966	Montana	WITHHELD		
	New Mexico	WITHHELD		
1967	Montana	WITHHELD		
	New Mexico	WITHHELD		
1968	Montana	3807	842	--
	New Mexico	6729	—	--
1969	Montana	775	—	--
	New Mexico	4855	—	--
1970	Montana	512	—	--
	New Mexico	4225	—	--
1971	Montana	142	—	—
1972	Montana	578	—	--
1973	Montana	239	—	—
1974		NO PRODUCTION		
1975		NO PRODUCTION		
1976		NO PRODUCTION		
1978		NO PRODUCTION		
1979		NO PRODUCTION		
1980		NO PRODUCTION		

platinum-group mineral production from stratiform or layered basic and ultrabasic intrusives, such as the Bushveld Complex have grown modestly over the last 20 years. These deposits are less affected by sharp swings in market price or level of demand.

Over the last five years the price of platinum has moved dramatically with the price of gold, rising from levels close to $250.00 an ounce in 1977 to over $500.00 an ounce in 1981. The price has declined since that time, falling back to levels between $300.00-$400.00 an ounce. Although rapid and volatile changes of gold and platinum prices can occur in a relatively short period of time, the expected growth trend for platinum prices is suggested to be steady, slow and upward, rising essentially at the average inflation rate of the industrial societies.

FIG. 4-5. TOTAL U.S. PRODUCTION OF PLATINUM

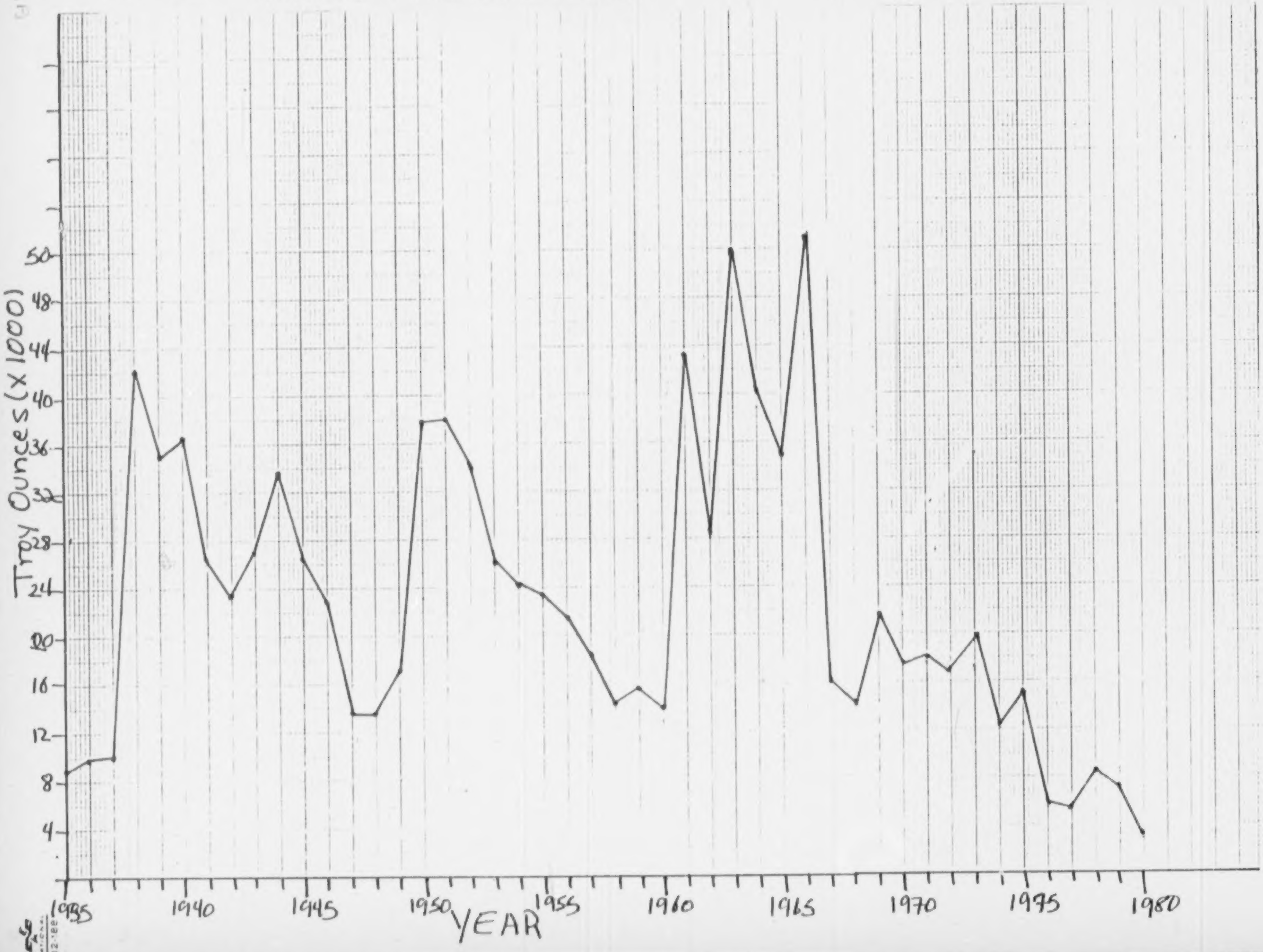


Table 4-20. Breakdown of Platinum Production by State and Region (troy ounces)

Alaska = Placer
California = Au byproduct

Year	State	Shipped	Region
1935	Alaska	8658	Goodnews Bay District
	California	293	Merced, Sacramento, Stanislaus, Yuba Counties
	Oregon	?	Curry County
1936	Alaska	8825	Goodnews Bay Distric, Koyuk, Ketchikan Districts
	California	900	Merced, Sacramento, Stanislaus, Yuba Counties
	Oregon	60	Curry County
1937	Alaska	9997	Goodnews Bay, Koyuk Districts
	California	452	Merced, Sacramento, Stanislaus, Yuba Counties
	Oregon	3	Curry County
1938	Alaska	41000	Goodnews Bay, Koyuk Districts
	California	1000	Merced, Placer Sacramento, Shasta, Tehama, Trinity Yuba Counties
	Oregon	43	Curry County
1939	Alaska	33900	Goodnews Bay District, Koyuk District
	California	1140	Merced, Placer, Tehama, Sacramento, Shasta, Stanislaus, Trinity Counties
	Oregon	20	Curry County
1940	Alaska	36000	Goodnews Bay District
	California	1400	Amador, Merced, Placer, Sacramento, Shasta, Siskiyou, Stanislaus, Tehema, Trinity Counties
	Oregon	69	Curry County
	Montana	31	Lewis and Clark County
1941	Alaska	25400	Goodnews Bay District
	California	1300	Amador, Butte, Merced, Placer, Sacramento, San Jouquin, Shasta, Stanislaus, Trinity, Yuba Counties
	Oregon	80	Baker, Grant, Jackson, Josephine Counties
	Montana	41	Lewis and Clark, Sanders Counties
1942	Alaska	22000	Goodnews Bay District
	California	1200	Butte, Calaveras, Eldorado, Merced, Humbolt, Placer, Sacramento, San Joaquin, Siskiyou, Yuba, Trinity, Stanislaus Counties
	Oregon	80	Grant County
	Montana	41	?

Table 4-20 (Page 2)

Year	State	Shipped	Region
1943	Alaska	27100	Goodnews Bay District
	California	62	Yuba County
1944	Alaska	33616	Goodnews Bay District
	Montana	9	Lewis and Clark County
1945	Alaska	26505	Goodnews Bay District
	California	43	Yuba, Trinity Counties
	Oregon	3	Josephine County
1946	Alaska	22882	Goodnews Bay District
	California	324	Butte, Sacramento, Shasta, Yuba, Siskiyou Counties
1947	Alaska	13512	Goodnews Bay District
	California	324	Butte, Merced, Sacramento, Yuba Counties
1948	Alaska	?	Goodnews Bay District
	California	?	Butte, Merced, Yuba, Sacramento, Siskiyou, Stanislaus Counties
1949	Alaska	?	Goodnews Bay District
	California	?	Butte, Merced, Yuba, Sacramento, Siskiyou Counties
1950	Alaska	?	Goodnews Bay District
	California	?	?
1951	Alaska	?	Goodnews Bay District
	California	?	?
1952	Alaska	?	Goodnews Bay District
	California	?	Butte, Yuba, Sacramento, San Matee Counties
1953	Alaska	?	Goodnews Bay District
	California	?	Sacramento, Yuba Counties
1954	Alaska	?	Goodnews Bay District
	California	?	Sacramento, Yuba Counties
1955	Alaska	?	Goodnews Bay District
	California	?	Sacramento, Yuba Counties
1956	Alaska	?	Goodnews Bay District
	California	?	Sacramento, Yuba Counties

55C

Table 4-20 (Page 3)

Year	State	Shipped	Region
1957	Alaska	?	Goodnews Bay District
	California	?	Sacramento, Yuba Counties
1958	Alaska	?	Goodnews Bay District
	California	?	Yuba County
1959	Alaska	?	Goodnews Bay District
	California	?	Sacramento, Yuba Counties
1960	Alaska	?	Goodnews Bay District
	California	?	Sacramento, Yuba Counties
1961	Alaska	?	Goodnews Bay District
	California	?	Sacramento, Yuba Counties
1962	Alaska	?	Goodnews Bay District
	California	?	Sacramento, Yuba Counties
1963	Alaska	?	Goodnews Bay District
	California	?	Sacramento, Yuba Counties
1964	Alaska	?	Goodnews Bay District
	California	?	Sacramento, Yuba Counties
1965	Alaska	?	Goodnews Bay District
	California	?	Sacramento, Yuba Counties
1966	Alaska	?	Goodnews Bay District
	California	?	Sacramento, Yuba Counties
1967	Alaska	?	Goodnews Bay District
	California	?	Sacramento, Yuba Counties
1968	Alaska	?	Goodnews Bay District
	California	?	Sacramento, Yuba Counties
1969	Alaska	21586	Goodnews Bay District
1970	Alaska	17385	Goodnews Bay District
1971	Alaska	18029	Goodnews Bay District
1972	Alaska	17112	Goodnews Bay District
1973	Alaska	19980	Goodnews Bay District
1974	Alaska	12657	Goodnews Bay District

Table 4-20 (Page 4)

Year	State	Shipped	Region
1975	Alaska	18920	Goodnews Bay District + Au Byproduct
1976	?	6116	Au Byproduct
1977	?	5545	Au Byproduct
1978	?	8246	Au Byproduct
1979	?	7300	Au Byproduct
1980	?	3348	Au Byproduct

Chapter 4

BIBLIOGRAPHY

Charles River Associates, 1982, Survey of the Cobalt Supply and Demand Picture (by B.J. Reddy). Paper presented at National Materials Advisory Board Committee Meeting on Technological Alternatives to Cobalt Conservation. Boston: Charles River Associates.

Dames & Moore Minerals and Resources Economics Group, 1980, Cobalt, Chromium, and Nickel Supply, Demand, and Strategic Importance. San Francisco: Dames & Moore.

Fischman, L.L., Project Director, 1980, World Mineral Trends and U.S. Supply Problems. Research Paper R-20. Washington, D.C.: Resources for the Future.

U.S. Bureau of Mines, 1982, Mineral Commodity Summaries, Washington, D.C.: U.S. Department of the Interior.

U.S. Bureau of Mines, 1983, Mineral Commodity Profiles: Nickel. Washington, D.C.: U.S. Department of the Interior.

Chapter V

CURRENT MINING ACTIVITIES AND PROPOSALS AND EVALUATION OF ISSUES AFFECTING DEVELOPMENT

This chapter summarizes the current status of selected potential mining projects in the United States where production of chromium, cobalt, manganese, nickel, or platinum could be expected. The projects selected for discussion in this chapter are those with a current or recent history of production or those where at least conceptual mining plans and associated environmental analyses are available for review. The goal of this chapter is to provide the reader with an understanding of (1) important aspects of the proposed mining and reclamation; (2) the environmental and socioeconomic setting and mining history of the project area; and (3) a summary of regulatory and environmental issues important to each project's future. This chapter thus serves as background for later chapters which analyze future production scenarios and the role of regulatory and environmental issues in affecting those production expectations.

Summary of Status of Various Projects

Only one operating mine in the United States produces any of the materials of interest to this study. This mine is the Goodnews Bay placer platinum mine along the Bering Sea coastline of Alaska; it has operated intermittently since 1934. The Nickel Mountain mine at Riddle, Oregon, operated from 1954 until April 1982 and still holds a valid mining and reclamation permit with the state of Oregon. The mine could reopen in a relatively short time if markets were found. The Blackbird cobalt mine west of Salmon, Idaho, has been evaluated and a final environmental impact statement has been published on the project. Noranda Mines, the operator of the Blackbird mine, has not, however, completed planing or siting of a smelter/refinery complex. Such a facility is necessary in order to develop the project, and the overall project not only awaits a sufficient market base, but also smelter/refinery development. Anschutz Mining has obtained sufficient permits to initiate pilot cobalt operations at the abandoned Madison lead Mine at Fredericktown, Missouri. A smelter must still be constructed at this site.

A draft environmental statement has been published for Anaconda Minerals' proposed Stillwater platinum/palladium project southwest of Billings, Montana, and a final statement is expected in 1983. This proposal is the focus of substantial environmental concern about reclamation and project impacts. PGM Resources, a joint venture of the Manville Corporation and Chevron, is in the preliminary stages of planning a platinum/palladium mine in the same area as Anaconda Minerals' proposal and is considering acquiring or joint-venturing the Anaconda mine.

The Six Rivers National Forest and Del Norte County, California, have published a draft environmental statement for California Nickel Company's proposed Gasquet Mountain Mine, a mine which would produce cobalt, nickel, chromium, and manganese.

In the Duluth Complex of northeastern Minnesota, AMAX proposed a large-scale operation to recover nickel, copper, and cobalt in the 1970s. However, this project has been shelved, and AMAX has relinquished control of the property. No other projects in the United States have proceeded beyond the exploration stage.

Summary of Environmental Concerns at Existing and Proposed Projects

Of these projects, the most substantial environmental concerns surround the opening of the Gasquet Mountain Mine in northwestern California. Here, there is considerable concern about the ability to revegetate processed tailings and the adequacy of revegetation to control erosion in the high winter rainfall conditions of the area. The possibility of uncontrolled erosion, in combination with the proposed construction of a dam, has generated considerable concern among environmentalists about effects on the Smith River stream system, all but a small part of which is designated a Wild and Scenic River. The stream system is of very high quality and is an important spawning habitat for salmon and steelhead. Other concerns about the project include the air quality effects of a proposed coal-fired power plant and processing facility on the Class I airshed of Redwood National Park, located six miles away, and the effect of mining on several rare and endangered plants endemic to Gasquet Mountain. The project will be carefully followed by environmentalists, has already received national attention, and may ultimately be contested.

Mining, if it were proposed in the Duluth Complex of northeastern Minnestoa, would also receive substantial environmental review, which might alter mining and smelter siting proposals. The major issue in this area is proximity of mining and smelting to the Boundary Waters Canoe Area (BWCA), a Class I airshed and an area environmentally sensitive to acid mine drainage and acidic precipitation. Analysis completed by the state of Minnesota indicates that if a "no-impact" policy were followed for the BWCA, no smelter could be sited in the immediate vicinity of the Duluth Complex, and even mining would be precluded in areas closest to the BWCA. If such limitations

Table 5-1

Summary of Regulatory and Environmental Issues at Likely Projects

Existing or Recently Producing Operations	Potential Production Rate (PGM)	Estimated Reserves (PGM)	Environmental an Regulatory Consid
Goodnews Bay, Alaska	10,000 oz.	500,000 oz.	No significant regul extreme environmenal prevent year-round o
Nickel Mountain, Oregon	5,000 MT (ni)		Mining permit still
Proposed Operations			
Blackbird, Idaho	1800 MTY (Co)	35,000 MT	Recovery of 0.27% Co tailings not propose
Anaconda-Stillwater	175,000 oz/y	3.5 mil T oz.	Draft EIS published Strong citizen conce emulative impacts wi potential mines Concerns about taili siting, effects to w
Madison Mine	1080 MTY (Co)	9090 MT (22,045 MT resource)	No regulatory proble
Gasquet Mountain	900 MTY (Co) 8773 MTY (Ni) 12,364 MTY (Cr)	16,200 MT (Co) 157,909 MT (Ni) 222,546 MT (Cr)	Draft EIS issued 3/8 opposition from loca conservation organiz powerplant sited 6 m Class I airshed; pot into wild and scenic and endemic plants i uncertain potential tion success; untest technology needed; l
Duluth Complex, Minnesota			
Zone 1	1.5 MT (Ni) 0.03 MT (Co)		Mining generating du result in exceedance air standards in BWC unlikely
Zone 2	0.8 MT (Co) 0.01 MT (Co)		open pit mining gene likely to result in Class I air standard open pit mining unli

were actually implemented, the possible mining proposal of INCO at Spruce pit would be prohibited. Mining in other parts of the Complex might also be limited. Unfortunately 75% of the copper-nickel resource of the Duluth Complex is located in locations most needing environmental protection--areas draining into the BWCA.

Mining in the Stillwater Complex of south-central Montana will face substantial environmental review, but it is not likely that such a review will result in prohibition of a mine. Citizens of this area are very concerned about water resource effects of mining, as well as the overall impact to lifestyles in this agricultural area if mining is renewed. There is no indication, however, that approval of the presently proposed Anaconda Minerals' project will be significantly affected by these concerns or that the mine will be contested. The adverse response of citizens to a proposal by PGM Resources would be greater due to fears about mining at a rate probably two and one-half times the size of the Anaconda project; however, ultimate mine approval is still not expected to be affected.

The only remaining issues at the Blackbird Mine relate to mitigation of socioeconomic impacts to the Salmon area if mining commences. Noranda has already obtained most necessary mining permits. Citizen groups in this area do not plan to oppose the proposal.

Environmental and regulatory concerns are not a significant factor at Goodnews Bay, Nickel Mountain, or Madison. Table 5-1a summarizes the resource base and issues at these projects.

Environmental Considerations at Other Deposits

Mining development in southeastern Alaska, if viable deposits are ever identified, may be constrained by location within federal park or wilderness areas. The Brady Glacier deposit is a nickel-copper sulfide mineral deposit that also contains cobalt and platinum-group metals. Resources of this deposit have been estimated by Bundtzen, et al. (1982), to include 530,000 MT of nickel, 24 to 54 million pounds of cobalt, and 580,000 ounces of platinum-group metals. However, it is unclear that these estimates are based on a large enough analytical and drilling program to have a high degree of confidence in the estimate. This deposit is located within the Glacier Bay National Monument, an area where mining is prohibited. Until a few years ago, mining was considered an acceptable land use in the monument, but Congress has since changed land management criteria. The deposit is under an active glacier and could only be reached by access from nonglacier areas; however, no such acceptable entry location is available on patented land (table 5-1b).

The Bohemia Basin nickel-copper deposit on the north end of West Chicigof/Yakobi Island, which may include eight to nine million pounds of cobalt, may be affected by the designated wilderness on the island. The deposit itself was excluded from the wilderness. Inspiration Development dropped its exploration program of the property in September, 1982, however, which may indicate that industry does not now view this prospect as a viable deposit.

Development of any of the small laterite or ultramafic deposits in northern California or southern Oregon may intiate the same concerns now raised at Gasquet Mountain. Particularly in California and in the Smith River drainage, environmental concerns about revegetation, erosion, and impact on the Smith will be considered. One of the reasons the Gasquet Mountain project is being contested is the fear by some residents that construction of a mill will make smaller deposits economical to develop.

Table 5-1b

Long-Term Speculative Projects

	Resource	Issue
Brady Glacier, Alaska	530,000 MT (Ni) 10,900-24,500 MT (Co) 580,000 oz. (PGM)	Within Glacier Bay National Monument; mining currently prohibited

Placer chromite deposits have been identified along the beaches of southwestern Oregon and in Piedmont streams in Maryland, North Carolina, and Georgia. Since the entire coastal zone of Oregon is owned by the state and the southern portion is nationally renowned for its scenic value, it is doubtful that mining would ever be allowed in that area. The potential mining in the eastern U.S. would affect small streams, and the extent of citizen resistance is unknown.

Goodnews Bay

The Goodnews Bay platinum mine has been the largest producer of platinum-group metals in the United States. This placer mine is located along the Salmon River, 10 miles from the Bering Sea coastline of Alaska, and is 450 miles west of Anchorage. For the past four years, the property has been held by the R. A. Hanson Mining Company of Spokane. Although actual production data are unavailable for this period, the company has predicted a 10,000 ounce per year production rate (Bundtzen, et al., 1982).

No regulatory or environmental issues which affect the manner or timing of development are known to affect this property.

The Goodnews Bay mining district was discovered in 1926 by Walter Smith, an Eskimo prospector, and by 1934, several mining companies were operating on tributaries of the Salmon River. In 1937, an eight cubic-foot-capacity bucket-line Yuba dredge was installed by the Goodnews Bay Mining Company and, by 1975 when production stopped, 545,000 ounces of native metals, mostly platinum, had been removed. The largest production year was 1938, when over 38,000 ounces of platinum-group metals were extracted. During production years, about 65 employees worked at the mine.

Four years ago, R.A. Hanson Mining Company, a privately held company, purchased the equipment, campsite, and 3380 acres of claims for an undisclosed price. During 1980 and 1981, work at the mine focused on refurbishing camps, machinery, and drilling equipment. The dredge was operated for part of each year, "but not at a profit" (Bundtzen, et al. 1982). The dredge was inadevertently sunk in 1982 but has since been raised. Maximum digging depth of the dredge is 60 feet and access to deeper deposits has necessitated overburden removal by dragline. The mining season extends from spring (March-April) until fall (October-November) and the camp is abandoned in winter. Initial work each spring focuses on refurbishing equipment until the thaw in April (Kerwin, 1982).

Reserves of the property are at least 500,000 troy ounces of platinum-group metals, in 62.9 million cubic yards of gravel, but pay gravels are as deep as 210 ft, well beyond present mining depths, which have not exceeded 80 ft. The source of these placer deposits is "unquestionably" the nearby Red Mountain dunite, but the "mother lode" has never been found (Bundtzen, et al. 1982). Average grade at this deposit is 0.01 oz/ton (Carnes, 1981).

The camp is now equipped for up to 100 people with dining facilities, recreational equipment (including a bowling alley), machine shops, bunkhouses, and electric power generation. Reportedly, the company is considering wind-generated electricity at this remote camp which is not even connected by road to the nearby coast.

There are several difficulties operating this mine. The remoteness and climatic severity are reportedly difficult on employees and services. Major equipment can only be shipped by barge during the limited period when the Bering Sea is ice free. Thus, planning is critically important. Equipment, such as the 1937 vintage dredge, is old, necessitating continued maintenance.

Other difficulties include sorting of clay from the gravels, and reworking clay-rich tailings, where up to 40% of the metal values may have been lost during old mining. The gravel deposits are characterized by "clay balls" which are held on sorting screens and do not wash easily. The company reportedly also has an incomplete understanding of its reserve base and has problems sampling the tailings for platinum content.

As previously mentioned, the company does not know how it will recover deep reserves, but has considered rapid drilling techniques. R. A. Hanson Mining has a machinery division which may work on this problem (Kerwin, 1982).

The U.S. Bureau of Mines (Carnes, 1981) in its analysis of the property, proposed three possible mining systems for recovery of deep deposits:

1. For one portion of the mine, called the upper river area, a surface mining system consisting of two crawler cranes with draglines supported by crawler tractors with bulldozers is feasible. This system has been used successfully to mine platinum in the past from other Alaskan sites and could produce 1800 tons per day (tpd) of ore;

2. Continued use of the bucket-line dredge for reprocessing clay-material tailings that have oxidized sufficiently over the years to liberate residual platinum values, and could produce 6800 tpd of ore; and;

3. Utilization of a new type of underground continuous mining machine similar to those used in the coal industry could produce 3000 tpd. The USBM has expressed concern that cave-ins and flooding would render this approach uneconomic, however.

It is not known whether the USBM has discussed any of these options with the company.

The company is very hesitant to release information on its operation, where it sells its product, or what its long-range plans are (Kerwin, 1982).

Nickel Mountain

Hanna Mining Company operated the Nickle Mountain Mine, four miles west of Riddle, Oregon, from 1954 until April, 1982. During that time, it was the only nickel mine in the United States. The mine and associated smelter are presently closed, although ore reserves have not been depleted. The company has not offered a timetable or economic scenario under which it would resume production.

Mineralization has been long known at Nickel Mountain. In 1882, 3000 tons of ore averaging 5% Ni was mined (Ridenour, 1978). Hanna Mining acquired the leases of the area in 1947.

According to Oregon regulatory personnel, the permit Hanna held for mining prior to cessation of operations is valid whenever the company chooses to reopen.

Geology

The upper 500 to 1200 ft of Nickel Mountain is underlain by relatively unserpentinized peridotite (harzburgite) and dunite. This peridotite is underlain by sheared serpentinite, sheared lenses of sediments and volcanic rocks of the Dolthan Formation (Jurassic age), or by the Dolthan Formation. The overall structure of the mountain is not well understood, but it appears that the peridotite is an erosional remnant of an overthrust sheet, and the Dolthan Formation is somewhat younger than the peridotites (Figure 5-1).

The ore is classified into three groups: soil, saprolite, and boxwork. The total area of various ore deposits is about 790 acres. The thickness of deposits varies from five to 200 feet. The profile of soil and weathered bedrock in the main Nickel Mountain ore body is shown in Table 5-1.

Soil accounts for about 20% of the reserve, averages 10 feet in thickness, and ranges from one to 40 feet in thickenss. The soil is rich in hydrous iron oxides and clay minerals. There is a gradational change from red to yellow soil reflecting the change from predominantly ferric to ferrous compounds. Nickel content in the soil varies from about 0.1% at the surface to 2% at relatively shallow depths in the yellow-brown soil.

Over 70% of the ore reserves are found in three types of saprolite distinguished on the basis of contained magnesium oxide (MgO): soft saprolite (15% to 25% MgO), hard saprolite (25% to 35% MgO), and saprolized peridotite (35% to 40% MgO). Mineralogy of the saprolite includes montmorillonite and serpentine, with lesser amounts of other minerals (Ramp, 1978).

Boxwork ores probably originally accounted for 10% of the total ore reserves, but much of this high-grade ore has already been depleted by mining. Boxwork is a chalcedony-rich ore having over 50% SiO2, and results from deposition by supergene processes along faults. These are the highest grade ores due to deposition of garnerite-group minerals including sepiolite and nickel-bearing chlorite. Specimens of garnerite ore may assay 20% Ni, but boxwork ores more commonly range from 1.5% to 2% Ni (Ramp, 1978).

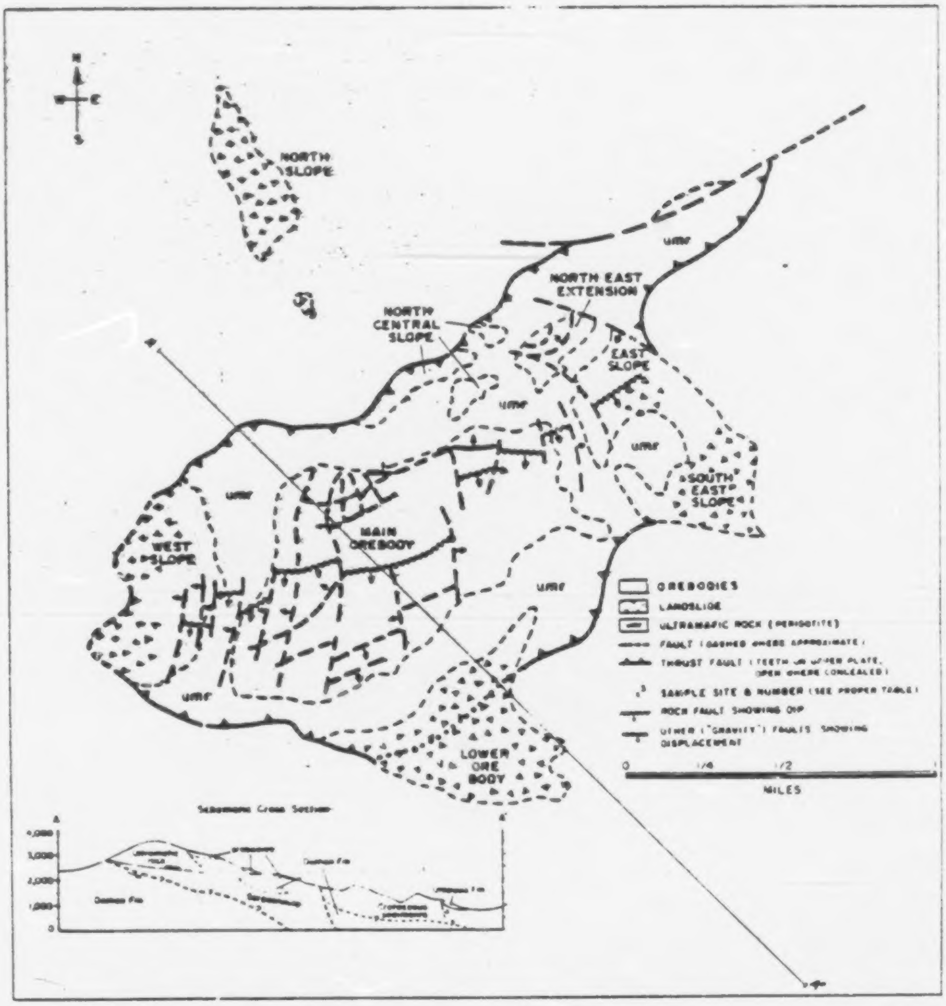

N
W
E
S
NORTH SLOPE
NORTH EAST EXTENSION
NORTH CENTRAL SLOPE
EAST SLOPE
SOUTH EAST SLOPE
WEST SLOPE
MAIN OREBODY
LOWER ORE BODY
umr
OREBODIES
LANDSLIDE
ULTRAMAFIC ROCK (PERIDOTITE)
FAULT (DASHED WHERE APPROXIMATE)
THRUST FAULT (TEETH ON UPPER PLATE, OPEN WHERE CONCEALED)
SAMPLE SITE & NUMBER (SEE PROPER TABLE)
ROCK FAULT SHOWING DIP
OTHER ("GRAVITY") FAULTS SHOWING DISPLACEMENT
0
1/4
1/2
1
MILES
A
A'
4,000
3,000
2,000
1,000
0

Figure 5-1 Geologic map and cross section of Nickel Mountain deposit showing upper peridotite thrust plate and outlines of orebodies. (Modified from Rowe and others, 1976)

Table 5-2 Classification characteristics of ore types at Nickel Mountain Mine
(Modified from Rowe and others, 1976)

Group	Ore type	Classification characteristics	Approximate Ni content (%)
Soil	Red soil	Fe - 12%; soil texture; brick-red color	0.1-0.8
	Yellow soil	Fe - 16%; soil texture; yellow to brown color	0.6-1.5
Saprolite	Soft saprolite	MgO - 24%; saprolite texture; less than 20% fresh peridotite	1 -2.5
	Hard saprolite	MgO - 24-34%; saprolite texture; 20-50% fresh peridotite	0.8-1.5
	Saprolized peridotite	MgO - 34-40%; saprolite texture; 50-75% fresh peridotite	0.5-1
	Peridotite	MgO - 40-45%; saprolite to fresh peridotite texture	0.2-0.3
Boxwork	Boxwork	SiO_2 - 50% (veins also present in saprolite); boxwork veins, massive chalcedony veins, or chalcedony bands; more than 75% fresh peridotite	1.5-5+

Mining

Ore was extracted at Nickel Mountain by open-pit techniques, where open-cut level benches have maximum 20-foot vertical intervals and minimum 50-foot widths, with benches connected by haulage roads. Most of the ore was dug without blasting by diesel shovels (3.5 and 5.5 yd^3). Hauling by 60-ton diesel trucks was from the pit to the screening and tram-loading plant. In the latter years of mining, about half of the mined rock were rejected. In 1978 daily ore and waste production was about 15,000 tons, which included boulders, screen oversize, and overburden. Of this 15000 tons, 3000 tons was rejected immediately as waste in the form of boulders and screen oversize, along with 5000 tons of overburden (Ridenour, 1978). Nickel production has averaged somewhat less than 12 million tons per year, however, maximum production has been estimated by the company to be 15 million tons (Johnson, 1982). Moisture content of the ore was about 20% freewater plus 8% water of crystallization.

Ore was transported down the mountain to the smelter on an aerial tram, a distance of 8300 feet and vertical drop of 2000 feet. The tram had maximum capacity of 250 short tons per hour and utilized 50 cubic-foot cable suspended tram cars.

In the first few years of production, the average grade of smelter feed was 1.5% but by 1977, grade had reportedly dropped to 1%. The final smelter product was ferronickel containing 50% Ni.

Production began at Nickel Mountain in 1954 when the government contracted to purchase 125 million pounds of nickel from Hanna Mining. Hanna Mining stated, in interviews, that production was stopped at Riddle for the following reasons:

1. Excess inventory (six to eight month supply at mine);

2. Depressed prices (the price of nickel fell from over $3.00 to less than $2.00 per pound);

3. Depressed demand, particularly in the steel industry;

4. Increased costs, particularly electricity; and

5. Decreasing ore grade.

Riddle is at a competitive disadvantage in the world nickel market because it mines the lowest grade ore of any operating world mine (Johnson, 1982).

Blackbird Project

The Blackbird Mine is located in Lemhi County, Idaho, about 13 miles south of the Salmon River and 21 miles west of the county seat of Salmon (Figure 5-2). Noranda Mining, Inc., a wholly-owned subsidiary of Noranda Mines Limited, is proposing to reopen the mine and to rehabilitate the existing concentrator, both of which have been closed since 1967. Proposed production at the mine is 1200 tons per day of ore containing copper and cobalt. The company has already gained approval for a 300 tpd pilot program; however, the higher production rate has been determined by Noranda as the optimal rate in order to also finance a refinery to process the cobalt and copper concentrates produced by the mill. The U.S. Forest Service issued a final Environmental Impact Statement on the project in February, 1982, but Noranda Mining has delayed development of the project. Presently, only a skeleton workforce maintains a water treatment plant at the mine.

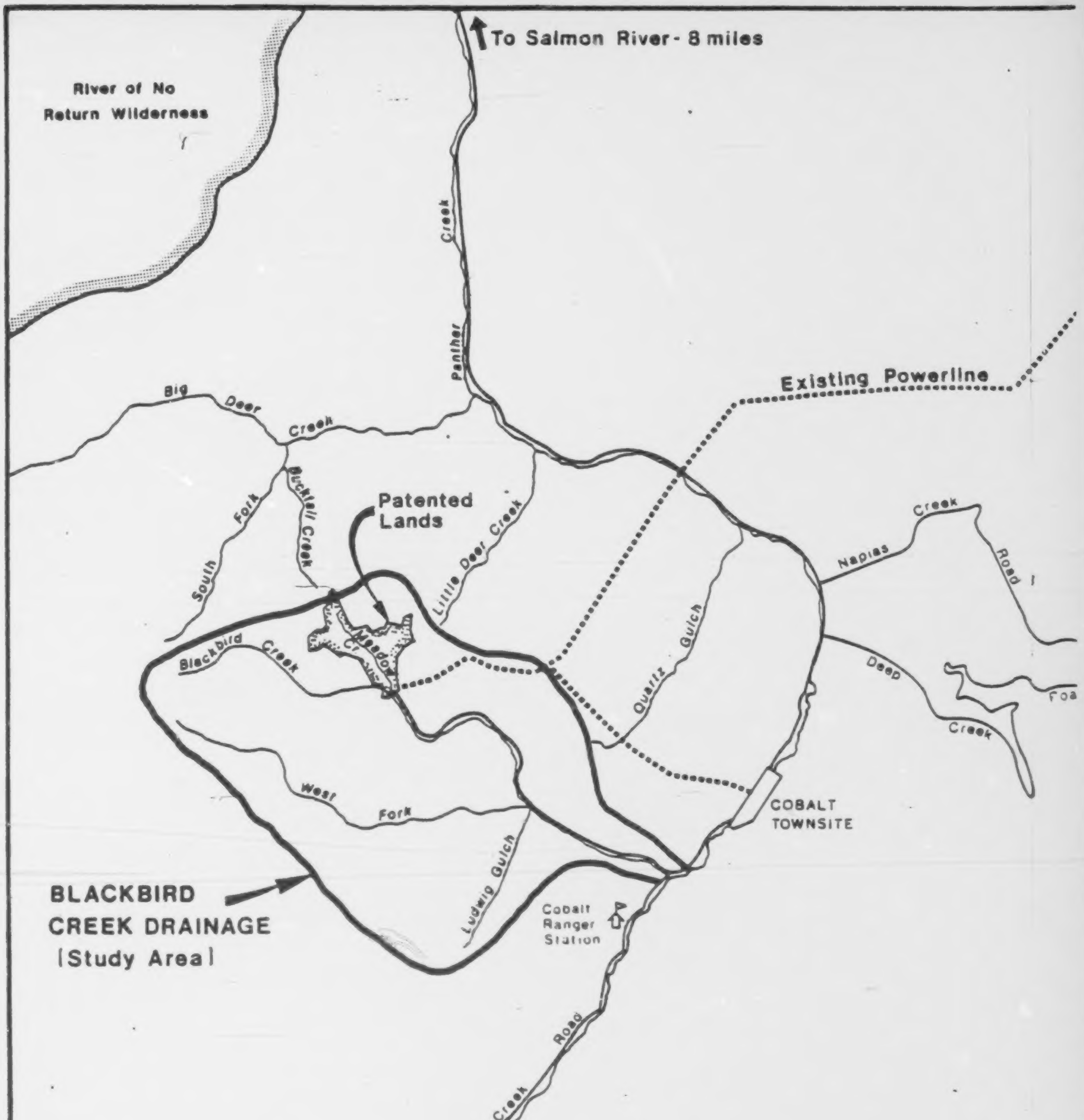

To Salmon River - 8 miles
River of No Return Wilderness
Panther Creek
Big Deer Creek
Existing Powerline
South Fork
Bucktail Creek
Patented Lands
Little Deer Creek
Napias Creek Road
Blackbird Creek
Quartz Gulch
Deep Creek
Foa
West Fork
COBALT TOWNSITE
Ludwig Gulch
BLACKBIRD CREEK DRAINAGE (Study Area)
Cobalt Ranger Station
Panther Creek Road
Panther Creek
Panther Creek
To Challis - 40 miles

Figure 5-2. Location of Blackbird Mine

Environmental Setting

The Blackbird project is in the Salmon River Mountains of central Idaho. In this area, streams have cut deep v-shaped canyons, typically about one mile wide and about 1000 feet deep. Slopes are very steep and rocky. Elevations range from 3000 feet on the Salmon River to peaks of over 9000 feet. Climatically, the area experiences a wet, mostly snowy, winter from November to March and a dry summer from July to September. Except for the valley floors and some north-facing slopes which are covered with spruce and fir, most of the area is forested with Douglas fir and lodgepole pine. These forests constitute a marginal timber resource because of the rocky, steep slopes of the area and extensive insect damage (Salmon National Forest, 1982).

Although wildlife diversity is high in Lemhi County, habitat in the Blackbird Creek drainage is generally poor due to low plant diversity (Becker, 1980). Large mammals are uncommon, and there is no elk or deer winter range in the Blackbird drainage. No threatened or endangered birds or mammals are known in the area, according to the U.S. Fish and Wildlife Service.

Virtually all of the project area is within the Blackbird Creek drainage basin, a tributary of Panther Creek. Panther Creek joins the Salmon River 25 miles downstream from the confluence with Blackbird Creek. Elevations in the Blackbird drainage range from 5500 feet at the mouth to about 7800 feet in the mine vicinity. The main access adit to the mine is presently at 6500 feet. Another tributary of Panther Creek, Big Deer Creek, drains a small portion of the Blackbird project area.

During the 1950's and 1960's, aquatic life in Panther Creek was virtually eliminated as a result of drainage from mining activity at Blackbird. Studies conducted in the area since 1967 show that dissolved copper concentrations have been the primary limiting factor controlling aquatic life in Panther Creek. Until the time Noranda began its pilot operations, Blackbird Creek had a pH of about 4.8 at its mouth and a copper concentration of 0.834 mg/l. The Federal criteria for aquatic life (EPA, 1980) are 0.0056mg/l of dissolved copper for a 24 hour average and 0.012 mg/l on a not-to-exceed basis. Quality of Blackbird Creek waters further upstream were even more degraded, and below the mill, the pH of the creek averaged 3.4. The poor quality has been the result of acid- and metal-laden discharges previously associated with waste piles and mine portals on Blackbird Creek and one of its tributaries, Meadow Creek. In Panther Creek, copper concentration increased dramatically below Blackbird Creek, declined steadily thereafter, then rose again at the confluence with Big Deer Creek (Fig 5-3).

In 1975, the Forest Service contracted a rehabilitation project to stablize 5800 feet of Blackbird Creek. Noranda, as part of its pilot program, installed a water treatment plant, and diverted a portion of Meadow Creek away from a large waste pile. The U.S. Forest Service estimates that these efforts have resulted in a 25% to 50% reduction in copper loading to Panther Creek.

The aquatic biology of the project area has been severely affected by these depressed water quality conditions. The Chinook salmon run dropped sharply following development of the mine in 1949. Extensive fish kills occurred in Panther Creek in 1954. The Idaho Fish and Game Department found no salmon redds, or spawning beds, in Panther Creek during periodic field checks made from 1968 to 1977 (Platts, et al. 1979). Blackbird Creek itself is essentially devoid of aquatic life. The Forest Service feels that Noranda's water treatment plant, however, is having a beneficial effect on the

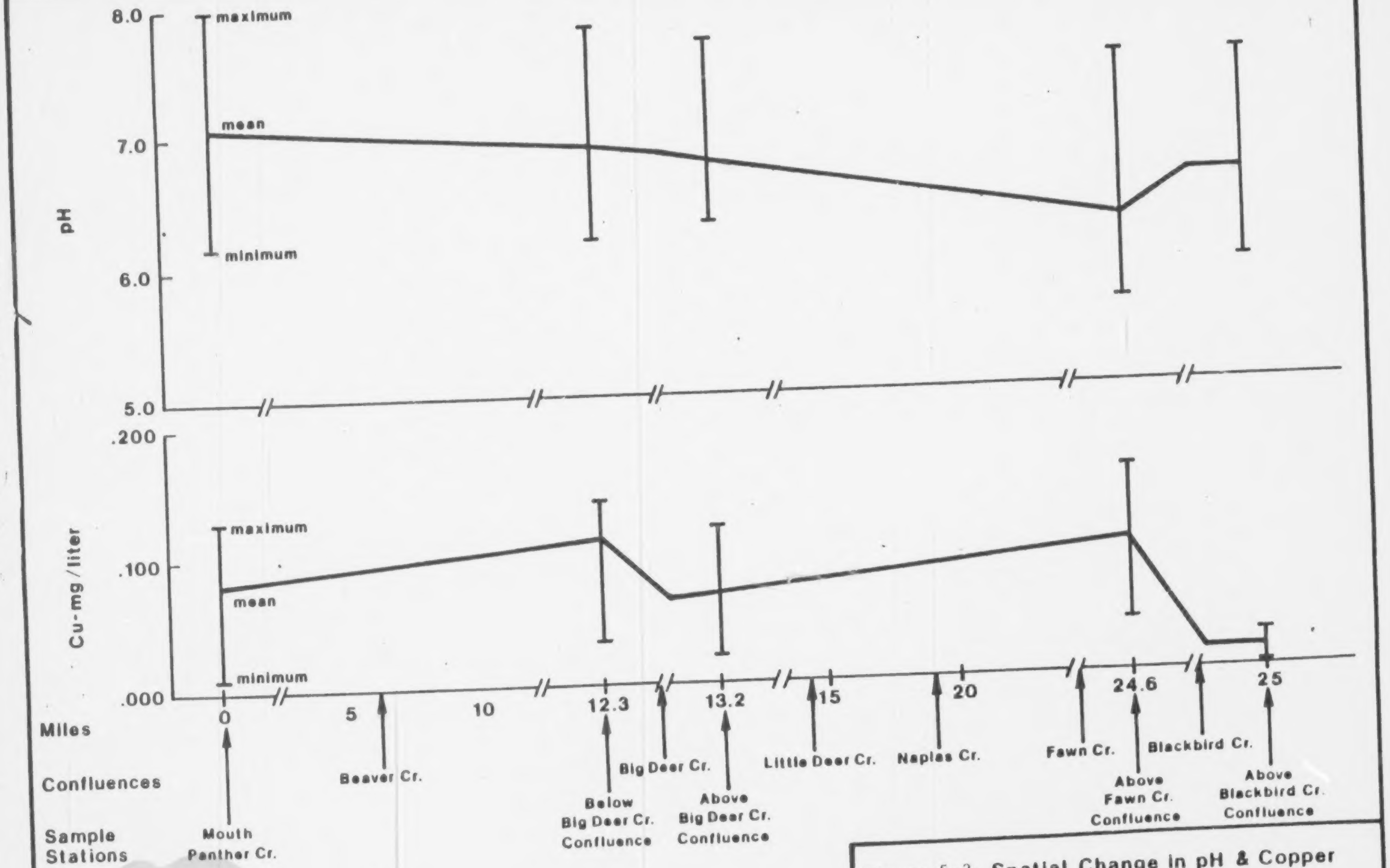


Figure 5-3. Spatial Change in pH & Copper Concentration in Panther Creek

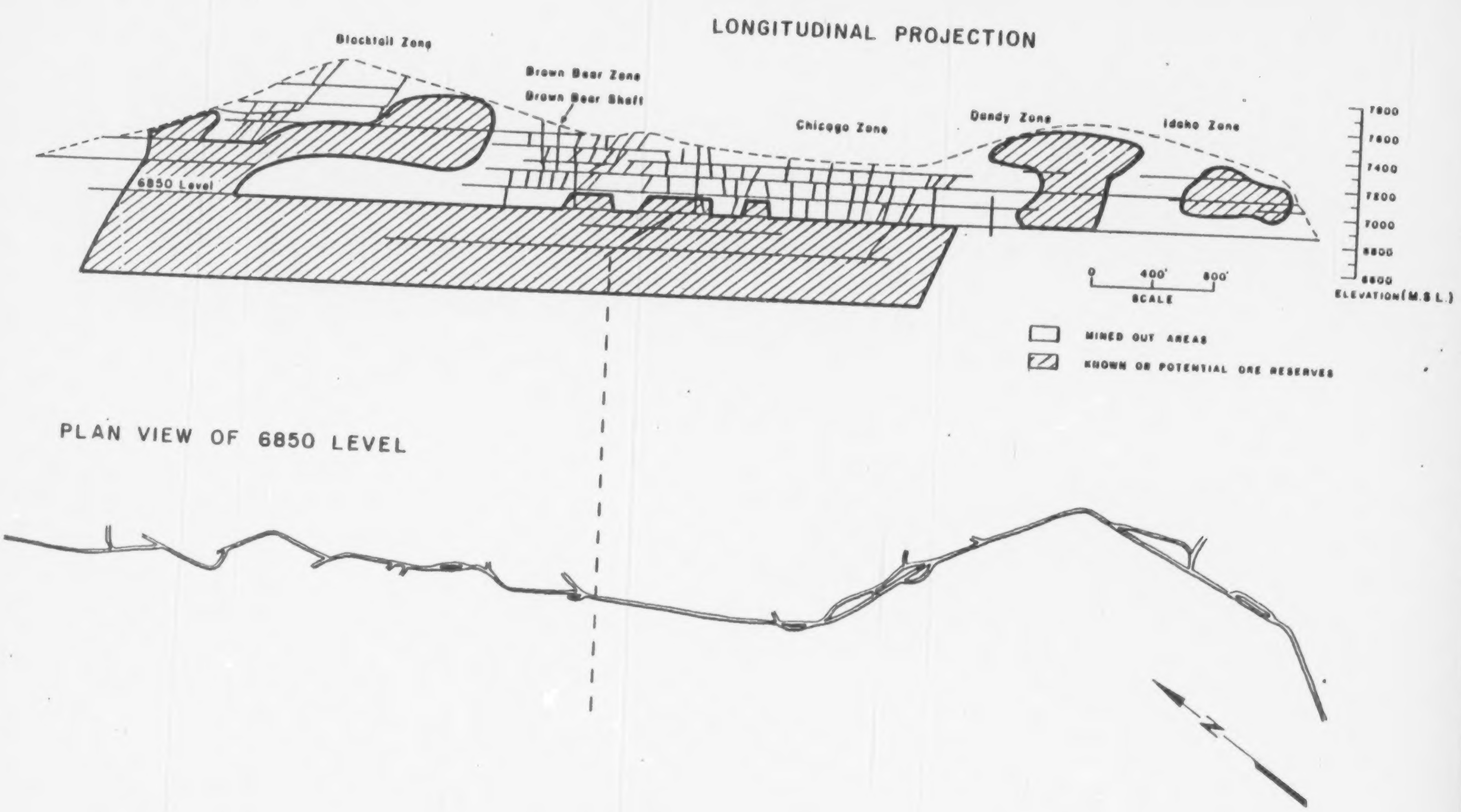
Source: ERT

fishery in Panther Creek, and a live salmon was even found in 1982 in Panther Creek (Buchta, 1982, pers. comm.).

Geology

The geology of the Blackbird area consists of slightly metamorphosed Precambrian quartzites (a granular metamorphic rock consisting mostly of quartz), pelites (mudstones), and biotite argillites (a rock derived from siltstone, claystone, or shale and which has undergone a slight degree of metamorphism). Thinly laminated and bedded micaceous quartzites predominate in the mine. Mineralization is highly variable in space, and continuity along strike and dip is poor. Ore bodies are irregular and vary from tabular to pod-like in form, from six to 70 feet in thickness. Along the two-mile strike-length of the mine, each (there are five) known ore body outcrops at the surface and dips northeast at about 50° to 60°. Ore varies from stratabound discordant to diffuse and each zone--Idaho, Dandy, Chicago, Brown Bear, and Blacktail--have unique geological and physical characteristics (Figure 5-4). Hydrous cobalt arsenate (eyrithrite) and copper carbonates (malachite and asurite) occur in surface outcrops. The main cobalt ore mineral is cobaltite (CoAsS) and the main copper ore mineral is chalcopyrite ($CuFeS_2$); both are sulfide minerals. Iron sulfides are widespread, particularly pyrite (FeS_2) and to a lesser extent pyrrhotite (FeS). There is no agreed upon model explaining mineralization at Blackbird (Peters, 1981).

Reported ore grades under Calera Mining and Hanna Mining were between 0.6% and 0.7% cobalt and 0.02% and 3% copper. Noranda reserves are between 0.5% and 0.6% cobalt and 1.5% copper. Substantial tonnage of lower grade material exists throughout the district. Rarely, ore blocks may approach 1.5% cobalt and 8% copper.



LONGITUDINAL PROJECTION
Blacktail Zone
Brown Bear Zone
Brown Bear Shaft
Chicago Zone
Dandy Zone
Idaho Zone
6850 Level
7800
7600
7400
7200
7000
6800
6600
ELEVATION (M.S.L.)
0
400'
800'
SCALE
MINED OUT AREAS
KNOWN OR POTENTIAL ORE RESERVES
PLAN VIEW OF 6850 LEVEL
N

Figure 5-4. Longitudinal Projection of Blackbird Mine and Plan of 6850 Level, Blackbird Mine, Lemhi County, Idaho

Estimates of reserves presently proposed for mining range between 25,000 to 35,000 tons of cobalt. The Salmon National Forest (1982) states that reserves are 25,000 tons of cobalt and 75,000 tons of copper. Bennett (1977) lists reserves as 35,782 tons of cobalt and 85,225 tons of copper in the positive, probable, and possible categories. Scheuering (1982) referred to 25,000 tons of demonstrated cobalt reserves and 10,000 tons inferred. Total ore reserves range between five and six million tons. Noranda has indicated that potential reserves at Blackbird could be 21 million tons (Fiorini, 1981). The company has relatively little data on reserves outside the presently proposed mining block. Reserves have only been estimated to the 6500 feet level. One deep hole, however, intercepted ore at the 6300 feet level (Scheuering, 1982). Presently proposed mine life is only 15 years although the tailings pond, as proposed, could accommodate 40 years of mining. Bilbrey (1962) estimated cobalt reserves could be as great as 112.2 thousand tons of cobalt.

The abandoned tailings impoundment on West Fork may prove to be a cobalt reserve at some future date because the upper 25 to 30 feet of the fill contains unrecovered cobalt. Hanna Mining carried this material as a reserve, at 0.27% cobalt. Noranda presently has no plans to recover the tailings, in part for fear of Noranda being held responsible for stability and reclamation of the abandoned dam (Scheurering, 1982). The tailings are not carried as a reserve by the company.

Historical Mining Activities and Current Development

Mineralization in the Blackbird Creek drainage has been known since 1893. Evidence of past mining activities includes existing buildings, waste rock piles, tailings impoundment, mine adit, and numerous roads. The

Blackbird ore body has been mined by both underground and open-pit methods. The open pit mining occurred in the late 1950's and left a 10.3 acre unreclaimed pit and approximately 3.8 million tons of waste rock deposited on 45.8 acres in the headwaters of Blackbird and Bucktail Creeks. About one million tons of waste rock were removed from the underground workings and open pit and placed in the drainages.

Tailings have been disposed of throughout the Blackbird drainage downstream of the concentrator. During the early years of mining, tailings were placed directly into the stream. In 1950, a tailings dam was constructed on West Fork Blackbird Creek (Figure 5-2) which now stores 2 million tons of tailings.

The first recorded cobalt production in the area occurred from 1917 to 1920 at the Haynes Stellite Mine, located about 1.5 miles southeast of the Blackbird Mine (55 tons of concentrate, averaging 17.8% cobalt). High transportation and refining costs reportedly forced closure of the mine. From 1938 to 1941, Uncle Sam Mining and Milling Company operated at the south end of the Blackbird Mine and produced gold, silver, and copper. Cobalt was considered undesirable, drew smelter penalties, and was not recovered.

World War II renewed interest in the area. Exploration work was carried out by Howe Sound Company, the U.S. Bureau of Mines, and the U.S. Geological Survey. Diamond drilling and trenching were undertaken and, in 1943, underground development of the Brown Bear and Chicago zones was begun by Calera Mining, a division of Howe Sound. From 1951 to 1956, development and exploration consisted of 41,000 feet of drifts and crosscuts, 20,000 feet of raises, and 75,000 feet of diamond drilling on eight levels. Building of the town of Cobalt was completed in 1952. Through 1957, all production was underground, and in that year about 2.5 million pounds of cobalt concentrate and 3.2 thousand tons of copper were produced. Most of Idaho's gold production also came from this mine. Over 400 men worked in the mine. In 1958, Calera opened the Bucktail open pit, and underground operations were suspended. Over 3.0 million pounds of cobalt concentrate were produced in that year (Bennett, 1977). The average throughout Calera's approximately ten years of operation was 800 to 900 tons of ore per day and in their last year of operation, 1959, they produced 1500 tons a day (Scheuering, 1982).

In 1960, the loss of contracts with the federal government, falling copper prices, the importation of cobalt from southern Africa and, as explained in Chapter VI, the inefficiency of the refining operation forced closure of the mine. Calera Mining wa liquidated and most of the equipment was sold to Machinery Center, Inc., who reopened the mine in 1966. Machinery Center operated the mill and leased individual sections of the mine to contractors for mining. In 1966, there were over 81,000 feet of workings in the mine (Figure 5-5).

Hanna Mining bought the property in 1967 and tried to run the mine and mill similar to the way Machinery Center had. However, Hanna Mining closed the mine after nine months and concentrated on exploration and development work. In 1979, Noranda Mining entered a limited partnership with Hanna Mining, with Hanna Mining being the limited partner and Noranda Mining being the operating partner. Essentially, the agreement states that after both parties recover their invested capital, Hanna Mining will maintain a 25% interest in the mine. Hanna Mining has an interest of $8 million in the property and Noranda Mining has spent about $40 million to date (Scheuering, 1982, pers. comm.).

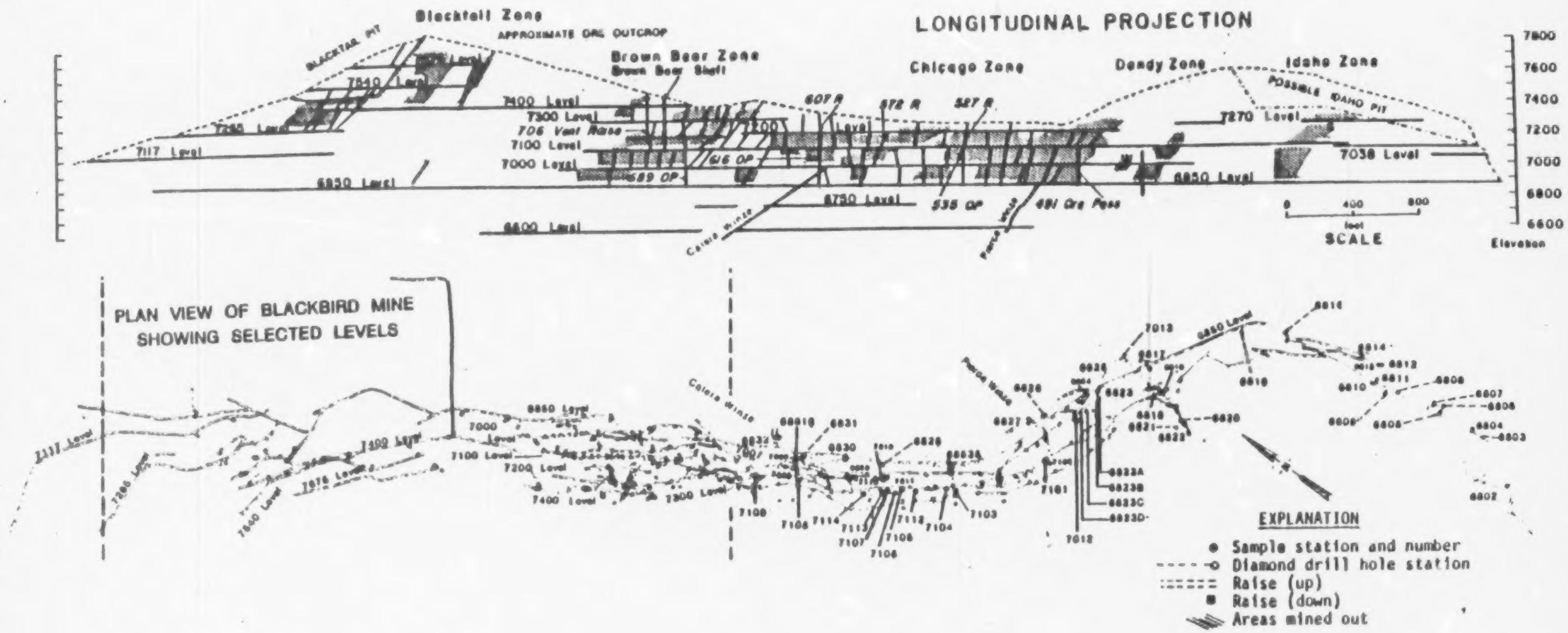

LONGITUDINAL PROJECTION
Blacktail Zone
BLACKTAIL PIT
APPROXIMATE ORE OUTCROP
Brown Bear Zone
Brown Bear Shaft
Chicago Zone
Dandy Zone
Idaho Zone
POSSIBLE IDAHO PIT
7840 Level
7250 Level
7117 Level
6930 Level
7400 Level
7300 Level
706 Vent Raise
7100 Level
7000 Level
689 OP
616 OP
607 R
572 R
527 R
7200
7270 Level
7038 Level
6850 Level
6750 Level
6600 Level
535 OP
481 Ore Pass
Catera Winze
0
400
800
feet
SCALE
7800
7600
7400
7200
7000
6800
6600
Elevation
PLAN VIEW OF BLACKBIRD MINE
SHOWING SELECTED LEVELS
7117 Level
7250 Level
7840 Level
7675 Level
7400 Level
7000
7100 Level
7200 Level
7300 Level
6850 Level
EXPLANATION
Sample station and number
Diamond drill hole station
Raise (up)
Raise (down)
Areas mined out

Figure 5-5 Plan view and logitudinal projection of the Blackbird mine (modified from Idaho Mining Co. (Hanna Mining Co.) mine map 1968?).

On September 8, 1980, the Forest Service approved Noranda's Plan of Operations for a pilot project to reopen the mine. Approval of this plan allowed Noranda to rehabilitate old mine facilities, construct some new ones, and mine cobalt at the rate of 300 tons per day. In addition to complying with Forest Service regulations, Noranda must meet the requirements of its National Pollutant Discharge Elimination System (NPDES) permit and a Compliance Schedule Order with the Idaho Department of Health and Welfare. The Compliance Order includes a clean-up program for Blackbird Creek within Noranda's private lands and a nonpoint pollution study of Blackbird Creek. The water treatment plant for the pilot operation was constructed to meet certain of these requirements.

The mill was operated from February to November, 1981, at a 30 ton per day rate. Metallurgical pilot tests were reportedly very encouraging. Mining and milling have been halted at Blackbird and only a skeleton workforce maintains the facility, awaiting a full production (1200 tpd) decision. The presently permitted 300 tpd operation is not considered an economically attractive option.

With approval of the final EIS by the Forest Service, the mine has cleared most regulatory hurdles prior to initiating full production. Once Noranda makes a full production decision, the company will have to negotiate a plan of operations with the Forest Service and obtain a discharge permit for the larger size mine. Neither issue is considered a stumbling block to mine development (Scheuering, 1982).

Noranda Proposal

Noranda proposes to develop an underground mine using cut-and-fill or undercut-and-fill methods, with a presently proposed mine life of 15 years. During preliminary mine planning, surface mining methods were considered; however, because most of the proven ore reserves occur as narrow veins at depths below 1000 ft, the stripping ratios are economically prohibitive. The friable rock conditions of the mine require a filling type method wherein the walls are stabilized. A two-circuit froth flotation milling process is proposed to recover cobalt and copper. A tailings pond which could store eight million tons of tailings is also proposed (40 years of mining).

Proven ore reserves have been located in five main zones. During the first few years of production existing mine workings will be used for access and haulage of ores. However, a new shaft will also be constructed. This approach will permit the earliest production while simultaneously constructing a shaft to access deeper reserves. Cobalt could be produced in about 18 months after a go-ahead decision was made (Scheuering, 1982, pers. comm.), although it is not known if mothballing the project for several years would increase this estimate. The company, however, usually estimates two to three years as its lead time, which includes construction of an associated refinery. The fill material used in mining will be a mixture of tailings and cement, pumped into the mine as a slurry from two backfill plants at the surface. After the new shaft and mine facilities are constructed and mine development proceeds below the 6600 foot level, rock from these lower levels will be hoisted out of the new shaft. Mine waters will be treated by hydrated lime neutralization and flocculation, with a capacity to treat 900 gallons per minute.

The presently proposed mining method is a fairly low productivity method, because ore bodies are generally 10 to 30 feet thick. One future possibility for Blackbird,* still very much under consideration, is development of a mechanized mining machine, specifically built for the dimensions of the Blackbird ore bodies. Productivity of the Noranda operation will be better than that achieved by Calera, which largely used a square-set, timber supported, stoping method. Timber support will be eliminated by Noranda by use of rockbolts and the undercut-and-fill method. Long-term mining options might include sublevel cave or blast-hole stope methods, but such methods necessitate acceptance of lower grade ores by the mill (Scheuering, 1982).

Figure 5-6 shows a schematic diagram of the flotation process described fully in Chapter VI. No new facilities are needed for crushing and screening; however, additional flotation circuits are needed for a 1200 tpd facility. Once the copper/cobalt concentrate is produced, it will be hauled by truck to an as yet unidentified and unconstructed refinery Noranda proposes to build in southern Idaho or northern Utah.

Noranda had considered many options for refining its cobalt concentrate. These options have included hauling concentrate to an existing refinery at Park City, Utah, and shipping the concentrate overseas. Neither option has proven economical, and the company sees no option other than construction of its own refinery. In analyses of critical components of the Blackbird project, the company has identified the refinery as the most important factor affecting viability of the entire project. In fact, the decision to develop a 1200 ton per day mine is based, not so much on optimal mine size, as on optimal production for delivery to a newly constructed and financed refinery (Scheuering, 1982). As discussed in Chapter VI, newly identified difficiencies in ore assays indicate that selenium is present in trace amounts in refined cobalt cathode. Noranda will now have to redesign

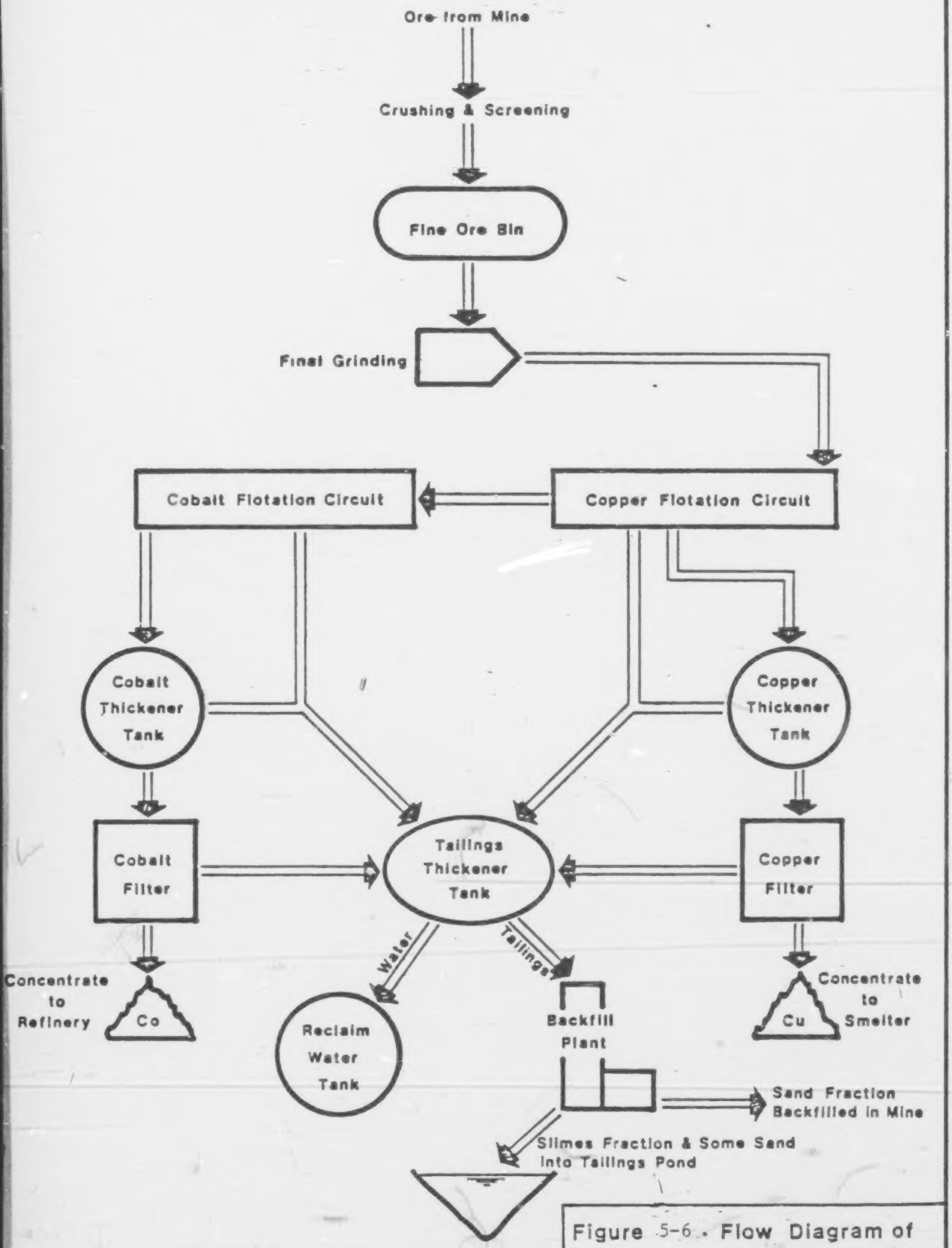

Ore from Mine
Crushing & Screening
Fine Ore Bin
Final Grinding
Cobalt Flotation Circuit
Copper Flotation Circuit
Cobalt Thickener Tank
Copper Thickener Tank
Cobalt Filter
Tailings Thickener Tank
Copper Filter
Water
Tailings
Concentrate to Refinery
Co
Reclaim Water Tank
Backfill Plant
Cu
Concentrate to Smelter
Sand Fraction Backfilled in Mine
Slimes Fraction & Some Sand into Tailings Pond

Figure 5-6. Flow Diagram of Flotation Process

Source: Noranda Mining Inc.

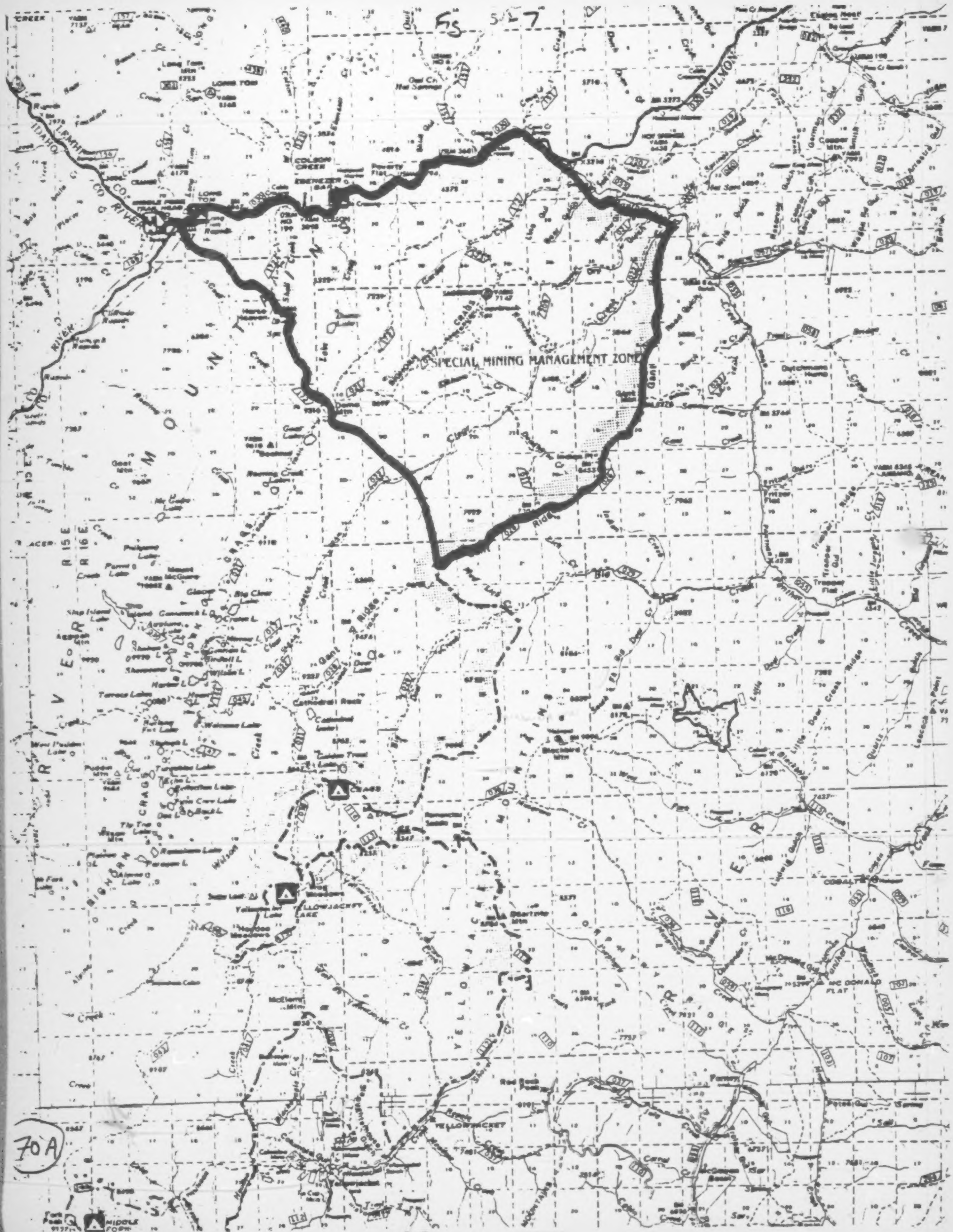
its process or develop another plan for eliminating the undesirable selenium.

Because of the swell factor and higher porosity of the settled backfill (tailings and cement mixture), only about 60% of the tailings produced by the concentrator will be backfilled into the mine. The proposed tailings dam would be a zoned earth and rockfill structure having a sloping upstream impervious core, a downstream rockfill shell, and a filter-transition zone between the core and the downstream shell. Upon completion of operation, the tailings impoundment would be covered with gravel and a clay seal to create a moisture barrier, and then covered with topsoil and revegetated.

Environmental analyses (Salmon National Forest, 1982) primarily focused on siting alternatives for the tailings impoundment. The Forest Service did not consider alternative mining or milling methods because the agency argued that the reopening of an old mine limits options. The preferred alternative recommended by the Forest Service is to construct the tailings facility and water supply reservoir in upper Blackbird Creek, above the mine facilities. This alternative was first proposed by Noranda and was supported by citizen comments to the draft EIS (Erickson, 1981). The Forest Service judges that long-term restoration and stability considerations are best met by locating both facilities in the upper watershed where soil considerations are best.

Citizen Concerns and Objectives

The Salmon Chapter of the Idaho Conservation League (ICL) has actively represented the interests of those citizens concerned about the environmental and social impacts of the Noranda proposal. The major concern about mine reopening is related to its effect on the City of Salmon and Lemhi County. In 1980, the population of Lemhi County was 7460, with 3303 in the City of


Fs 5-7
SALMON
LEMHI
RIVER
COLSON CREEK
EBENEZER
Poverty Flat
SPECIAL MINING MANAGEMENT ZONE
MOUNTAINS
BIGHORN CRAGS
CRAGS
Cathedral Rock
YELLOWJACKET LAKE
YELLOWJACKET
COBALT
McDONALD FLAT
Blackbird Mtn
MIDDLE FORK
RIVER OF NO RETURN
70A

78 B

Salmon. Major sources of employment in the County are agriculture (29%), trade, including a growing recreation sector (28%), and government (19%). The unemployment rate in Lemhi County in 1980 was 9.8% (Salmon National Forest, 1982).

Debates have arisen about the mine-induced population increases to the region. The Forest Service presently projects an increase of 1153, but other estimtes have reached 1925. The effect of this increase on social services, taxation, and land use has been of concern to many area residents. ICL has taken the position that Noranda should prepay taxes to Lemhi County and impacted school districts so that these entities can prepare for change. Noranda has yet to commit itself to a social impact mitigation program.

However, the ICL is not taking a position opposing the Blackbird Project. The ICL feels that the EIS did not adequately address alternatives in environmental or social impact mitigation, but the organization did not appeal the Final EIS. The organization is, however, opposing subsidies to Noranda.

River of No Return Wilderness

The Central Idaho Wilderness Act of 1980 (P.L. 96-312) established the River of No Return Wilderness, the boundary of which is within five miles of the Blackbird Mine. One of the controversies surrounding establishment of the wilderness involved the preemption of exploration and development rights for cobalt in roadless areas along the mineralized trend north of the Blackbird Mine. In response to these concerns, Congress established a Special Mining Management Zone for a 65 square mile area (Figure 5-7), known as the Clear Creek Area. In that zone:

> ...all prospecting and exploration for, and development or mining of cobalt and associated minerals shall be considered a dominant use of such land and shall be subject to such laws and regulations as are generally applicable to National Forest System lands not designated as wilderness or other special management areas. [PL 96-312(5)(d)(1)]

Within the Clear Creek area, no authority is given to the Forest Service to restrict mining based on provisions of the Wilderness Act of 1964. In the Conference Committee report on the Central Idaho Wilderness Act, the intent of this provision was explained: "There should be no doubt that the intent of the conferees ... was to remove any restriction based on the Wilderness Act on the mining of cobalt and associated minerals within the zone." Congress placed some restrictions on land use in the zone:

1. all mining locations and associated access can only be used for mining purposes or by the State of Idaho in order to manage bighorn sheep;
2. patents issued by the federal government convey only title to the minerals and not the land surface;
3. use of patented land can only be directly related to mining activity; and
4. the Secretary of Interior 'may take all reasonable measures to see that the mining or processing of cobalt and associated minerals ... does not significantly impair the overall habitat of the bighorn sheep.' [PL 96-312(5)(d)(1)(C)]

To date, the evidence of mineralization in the area is poorly known, and is limited to geochemical anomalies and surface outcroppings of mineralized rocks. Noranda Exploration, Inc., has received approval to undertake drilling in the Management Zone, using helicopters for access; however, a company cutback in exploration led to cancellation of the project for the 1982 field season (Cobalt Ranger District, 1982). In assessing the impact of the proposed exploration, the Forest Services' approval included several modifications of Noranda's original proposal. Modifications in helicopter access are intended to minimize impacts to wildlife, recreation, and visual resources.

GASQUET MOUNTAIN

California Nickel Corporation, a wholly-owned subsidiary of Ni-Cal Developments Ltd. of Canada, is proposing to open a surface mine a laterite deposit in in extreme northwestern California, near Crescent City (Figure 5-8). California Nickel holds 8000 acres of claims on Gasquet Mountain, on land administered by the Six Rivers National Forest. About 3000 of these acres would be mined during the estimated 18 year life of the project. Annually, the company proposes to recover 900 MT of cobalt, 8,773 MT of nickel, 45,455 MT of chromite concentrate (40% CR203), as well as magnesium oxide.

The project has been controversial from the time the company first submitted a draft environmental impact report to the Del Norte County Planning Commission in 1981. The Six Rivers National Forest and Del Norte County published their own draft environmental impact statement in March 1983. The viability of the project hinges on (1) the economics of multiple metal production and processing, and (2) the ability to mitigate several environmental impacts and to demonstrate reclamation feasibility.

Environmental Setting

Gasquet Mountain is located in the Klamath Mountains geologic province near its boundary with the Coast Range province. The terrain of the area is very complex, characterized by deep, narrow river valleys surrounded by 2000 to 3000 feet mountains, some of which have broad, flat crests. On these crests are found lateritic soils.

Climatologically, the most distinctive characteristic of the area is its precipitation. In the three years of data collection at the proposed mine site, annual rainfall has been 65 (1979), 67 (1980), and 134 (1981) inches

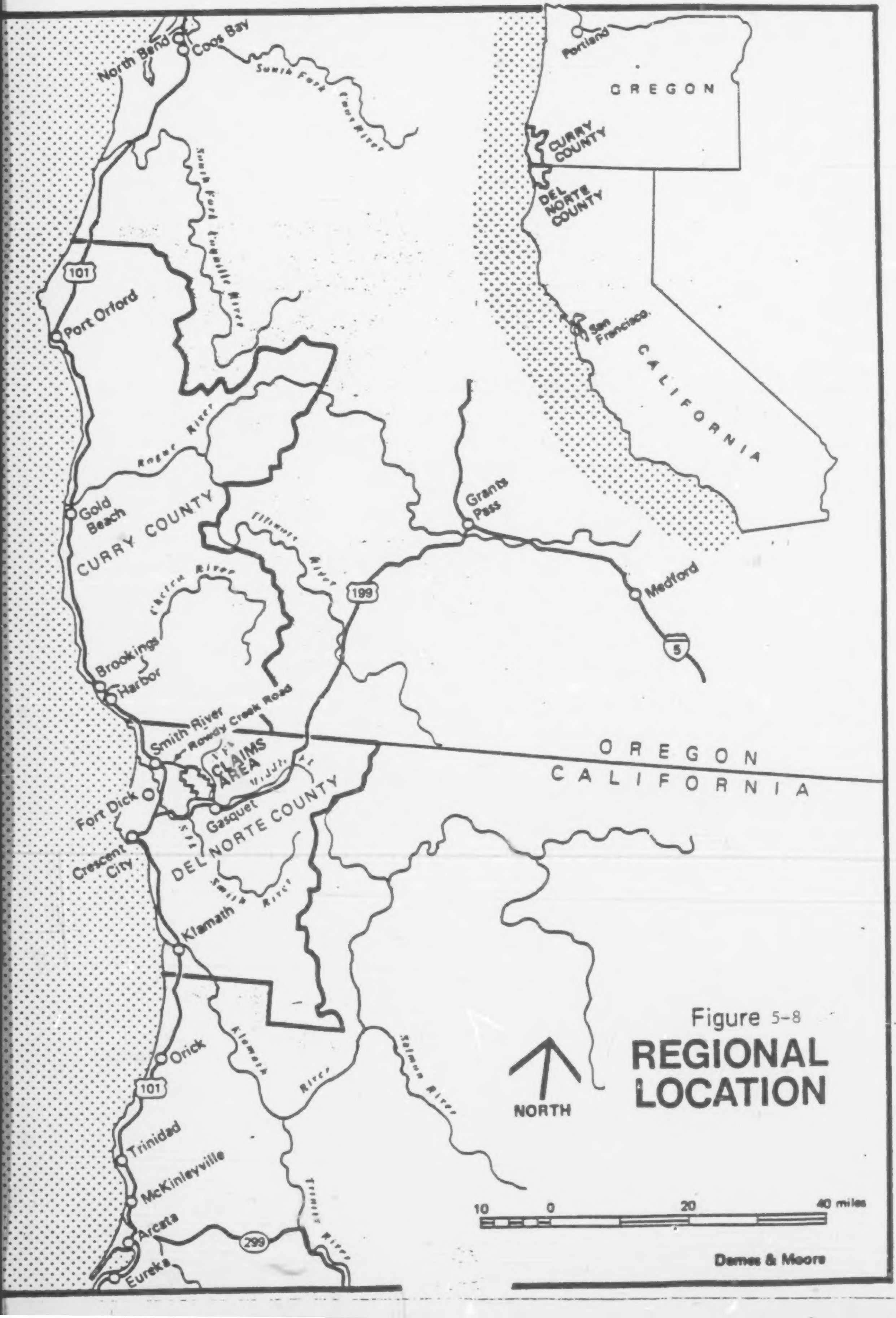

North Bend
Coos Bay
South Fork Coos River
South Fork Coquille River
101
Port Orford
Rogue River
Gold Beach
CURRY COUNTY
Chetco River
Illinois River
Brookings
Harbor
Smith River
Rowdy Creek Road
CLAIMS AREA
Fort Dick
Gasquet
Crescent City
DEL NORTE COUNTY
Smith River
Klamath
Orick
101
Klamath River
Salmon River
Trinidad
McKinleyville
Arcata
299
Trinity River
Eureka
Grants Pass
199
Medford
5
OREGON
CALIFORNIA
Portland
OREGON
CURRY COUNTY
DEL NORTE COUNTY
San Francisco
CALIFORNIA
NORTH
10
0
20
40 miles

Figure 5-8

REGIONAL LOCATION

Dames & Moore

80f3

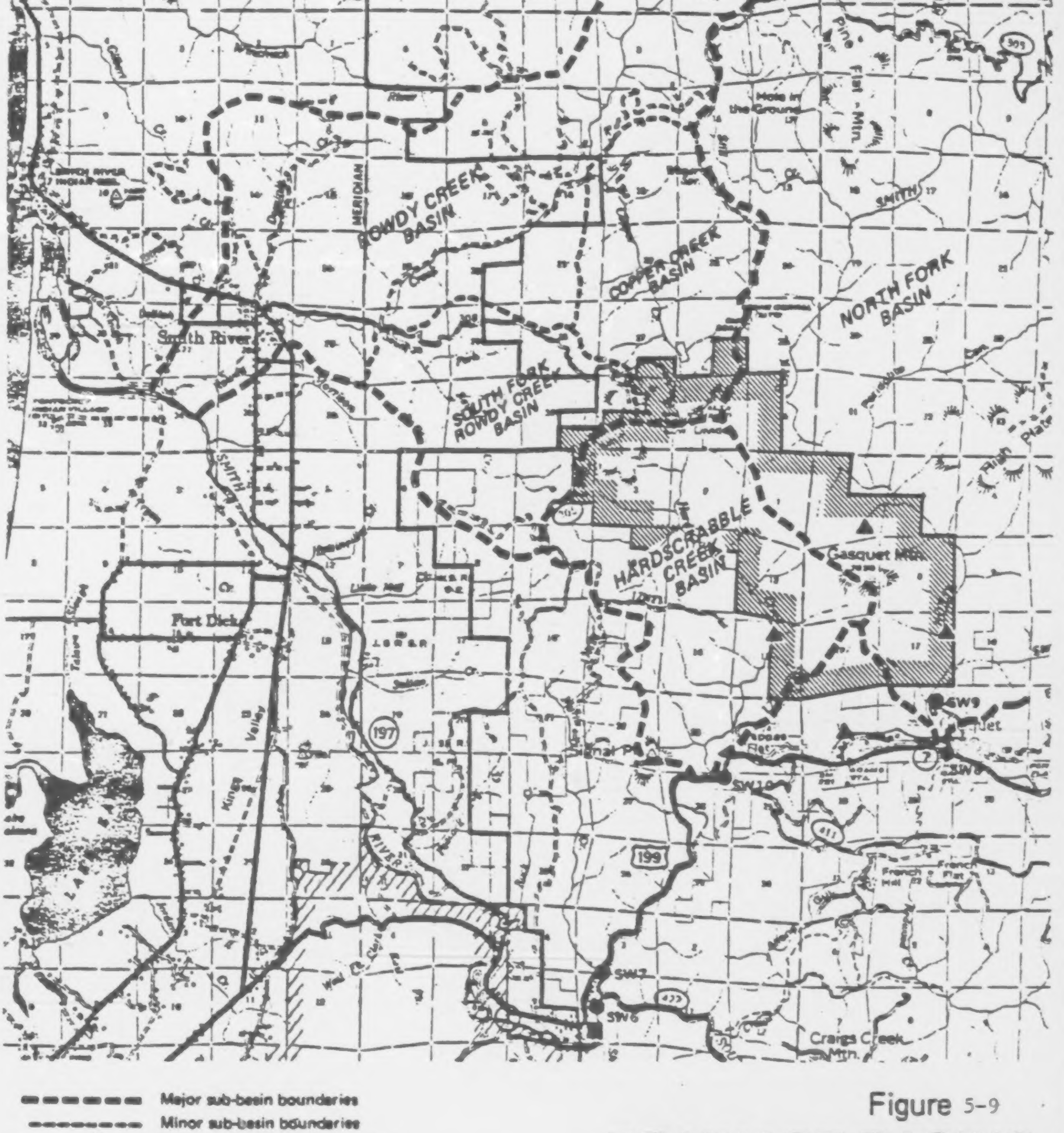
(U.S. Forest Service, 1983). Estimated total snowfall at Gasquet Mountain is about 15 inches. As is characteristic of the northern California coastal area, most precipitation occurs in the winter, and about 65% falls from November to February. Occasionally, the region experiences heavy, persistent rains over a period of 7 to 10 days. Mean annual temperature for the area is about 50°F.

The deeply dissected topography of the area tends to collect cold air that drains into the valleys. This condition results in either weak, morning inversion layers or more persistent inversions, often occurring in the fall and winter.

For purposes of administration of Prevention of Significant Deterioration (PSD) regulations consistent with the Clean Air Act of 1970, Del Norte County has been classified as an attainment area for all criteria pollutants, and Gasquet Mountain is within a Class II area. Redwood National Park, located six miles from the proposed plant site, is the closest Class I area. The California Air Resources Board has recently proposed Jedediah Smith and Del Norte Coast Redwood State Parks, located seven and 13 miles from the proposed plant site, to also be designated as Class I areas.

Gasquet Mountain drains to the Smith River. This river system, including all tributaries except Hardscrabble, Copper Hill, and Copper Creeks, has been designated wild or scenic under state and federal Wild and Scenic River statutes.

The western portion of the Gasquet Mountain project area is drained by Hardscrabble Creek, which joins the Smith River downstream of the town of Gasquet (Figure 5-9). The eastern part of the proposed project area is drained by the North Fork of the Smith River, which joins the Smith River at


SISKIYOU
NATIONAL
FOREST
CURRY
DEL NORTE
101
199
197
ROWDY CREEK BASIN
COPPER CREEK BASIN
NORTH FORK BASIN
SOUTH FORK ROWDY CREEK BASIN
HARDSCRABBLE CREEK BASIN
MERIDIAN
Smith River
Fort Dick
SMITH
RIVER
Hole in the Ground
Flat Mtn.
Gasquet Mtn.
Signal Pk
Kings Valley
SW9
SW10
SW8
SW7
SW6
French Hill
Craigs Creek Mtn.

- ▬ ▬ ▬ Major sub-basin boundaries
- ‒ ‒ ‒ Minor sub-basin boundaries
- ● SW6 Dames & Moore sampling stations
- ▲ Possible Dames & Moore sampling stations
- ① USGS stream gage, Smith River near Crescent City (drainage area 609 sq. mi.)
- ⑦ USGS stream gage, Middle Fork Smith River at Gasquet (drainage area 130 sq. mi.)
- ■ USGS sampling station (no. 11532500)
- ▨ Project area

1 0 1 2 3 4 miles

NORTH

Figure 5-9

DRAINAGE BASINS AND SURFACE WATER SAMPLING STATIONS

Source: USGS 15-minute quadrangles Crescent City and Gasquet

Base map from Six Rivers National Forest, U.S. Forest Service, 1977.

Dames & Moore

the town of Gasquet.

The Smith River basin exhibits one of the highest precipitation and runoff rates in the continental United States, exceeded only by a few basins in Washington and Oregon. Winter precipitation (October through April) produces about 90% of the annual runoff. The average annual runoff is 78% of the annual precipitation.

Excessive erosion and high stream sediment yields are problems throughout northern California. The potential for soil erosion in the Smith River basin is high due to its steep slopes, intense precipitation, and logging activities. On-site investigations at Gasquet Mountain, however, have indicated relative stability of the laterite soils, particularly at higher elevations. The primary causes of erosion in the area are soil slumping and landslides.

Water quality in the Smith River basin is high, with low total dissolved solids concentrations (less than 130 mg/l). Waters are alkaline (pH values from 7.3 to 8.2) and are of a magnesium-sodium-bicarbonate type. At present the waters of the Smith basin are used for recreation, irrigation, domestic, and municipal purposes. The town of Gasquet takes its water at the confluence of North Fork and Middle Forks Smith River.

The Smith River is a very important habitat for anadromous fish--salmon and steelhead--vital to the northern California fisheries economy. The lower portion of Hardscrabble Creek as well as the main Smith River are designated anadromous fish habitat.

In general, the soils of the Gasquet Mountain area are finely textured, shallow, and contain a high percentage of coarse fragments. For those soils occupying the largest part of the proposed mine area, permeability is moderate to rapid, internal drainage is good to excellent, and surface runoff is low to high. Erosion hazard on the in-place soils, following disturbance, is moderate to low, and observations made on road cuts of historically disturbed areas indicate little erosion.

The fertility of the laterite soils is radically different from that of most northern California soils. The Gasquet Mountain soils are typically high in magnesium and manganese and low in calcium, nitrogen, phosphorus, and potassium. Although soil chemistry is the primary cause of soil infertility, physical and characteristics such a shallowness, stoniness, fine texture, and excessive drainage, also present problems.

The vegetation of the Gasquet Mountain area likewise contrasts sharply with the dense forests of most of coastal northern California. Overall, the project area is a relatively open, poorly stocked forest with varying degrees of brushy, sometimes grassy undergrowth, alternating with expanses of chaparral. Surveys of the California Nickel claims area have identified eight plant species that could be classified as endangered under the Endangered Species Act and additional studies are being carried out to investigate the occurrence of an additional six candidate species which may also occur on the property (U.S. Forest Service, 1983).

The bald eagle and peregrine falcon, both officially listed as endangered, may occur in the vicinity of the claims. Peregrines, which are migrants and winter visitors, have historically nested in the area. One nest southwest of the claims was active in 1975, and is currently being observed for activity. If the nest is still in use, the Forest Service has proposed a management plan in which certain activities are controlled during the nesting period. One-half to two-thirds of the Gasquet Mountain claims area would fall in a zone where loud noises such as from helicopters or blasting would have to be controlled.

Socioeconomic Setting

The Gasquet Mountain project is located in a sparsely inhabited area in north-central Del Norte County. The site is approximately 15 miles northeast of Crescent City, the county seat. The town of Gasquet is the closest population center to the project, and had a population of 550 in 1980. The economy of the area has been described by the Del Norte County Planning Department (1975) as "a small, static lumber-exporting economy evidencing large cyclical swings, high unemployment and relatively low income." Fishing and tourism comprise other major sectors of the local economy, and there has been a recent trend of in-migration of semi-retired and retired persons into the area. In the current depressed economy, government is the largest employer in the county. Berthelsen (1982) described the current state of the economy in the area:

> Crescent City is not in a recession. It is in a depression. The town and surrounding Del Norte County are among the most economically depressed areas in California. Currently, 25% of the job force collects unemployment insurance. F.W. Godfrey of the Tri-Agency Economic Development Authority in Crescent said earlier this year that as many as 40% actually are out of work, but the benefits of some 15% have run out.

Almost a fifth of the households in the county, he said, are at least a month behind on their electric bills. (p. 56) The proposed Gasquet Mountain project is considered by many to be a potential saviour to the economy.

Geology

The Klamath Mountains, within which is found the Gasquet Mountain project, are composed of arcuate belts of sedimentary, volcanic, and plutonic rocks of Paleozoic to Cretaceous age, as well as their metamorphic equivalents. These arcuate belts have been interpreted as a series of thrust sheets (Burchfield and Davis, 1981), and the contacts between the belts are marked by faults. Gasquet Mountain is almost entirely underlain by intrusive ultramafic rocks of the Josephine peridotite formation. Locally, these rocks have been altered by serpentinization at varying degrees. The unaltered ultramafic rocks have been intensely weathered to iron-rich lateritic soils enriched in nickel, cobalt, and chromium.

Five main classes of bedrock are found at Gasquet Mountain (Figure 5-10): non-serpentinized harzburgite; partially serpentinized harzburgite; completely serpentinized harzburgite; dunite; and metavolcanic and metasedimentary rocks of the Galice formation. Three of these formations are of particular importance to understanding the geology and mineralization of Gasquet Mountain:

1. The non-serpentinized harzburgite usually develops a laterite soil profile during the weathering process and is thus the source material or ore of nickel, chromium, and cobalt to be produced by the proposed mining.
2. The partially serpentinized harzburgite usually does not develop a lateritic soil during the weathering process.
3. The completely serpentinized harzburgite (serpentinite) is usually located in shear zones, and does not develop a lateritic soil profile. Thus, it is not suitable ore material for the proposed mining activity (U.S. Forest Service, 1983).

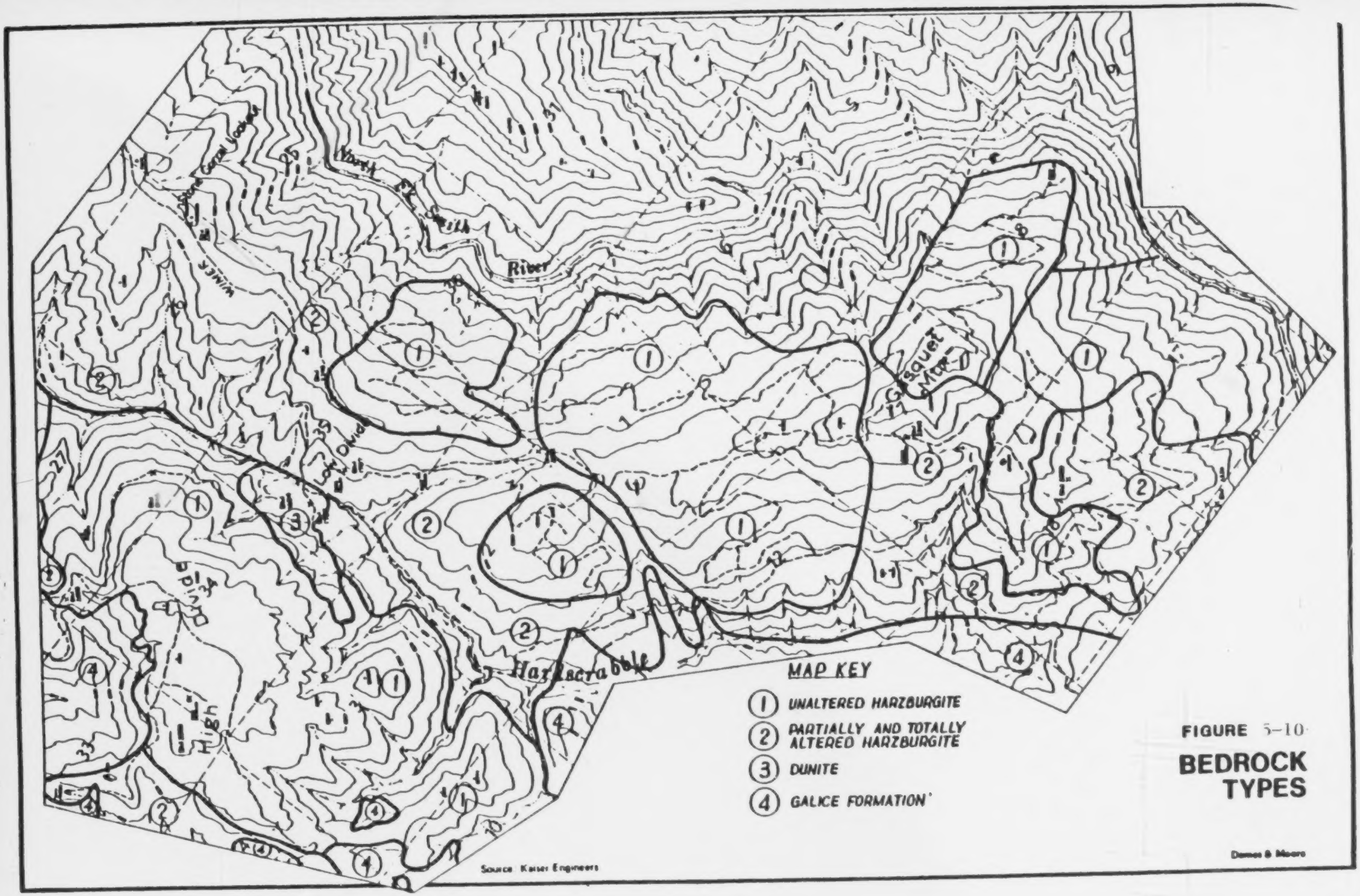

North Fk. Smith River
Gasquet Mtn.
Hardscrabble
MAP KEY
① UNALTERED HARZBURGITE
② PARTIALLY AND TOTALLY ALTERED HARZBURGITE
③ DUNITE
④ GALICE FORMATION

FIGURE 5-10

BEDROCK TYPES

Source: Kaiser Engineers

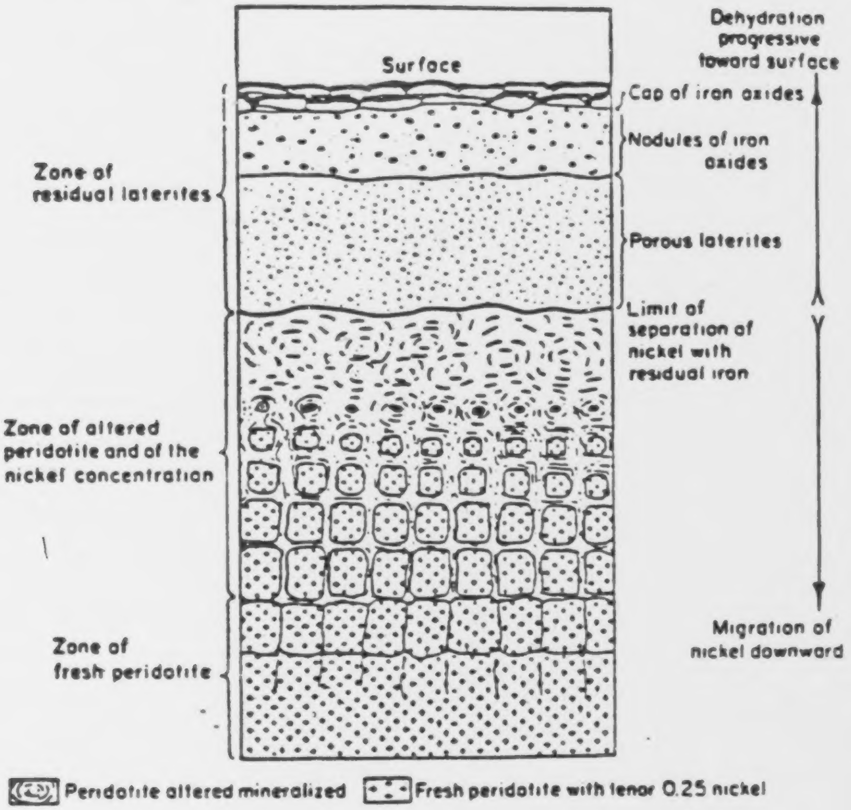
Dames & Moore

Therefore, the distribution of laterite, and also ore, on Gasquet Mountain is controlled by the distribution of peridotite and serpentinite lithologies. Areas underlain by relatively unaltered peridotite are typically rich in laterite soil and relatively serpentinized material is usually devoid of laterite.

Typically, the soil at the surface in laterite areas is a reddish-brown hematitic laterite, high in iron and cobalt with minor amounts of concretionary iron nodules (Figure 5-11). The thickness of this zone averages two feet, but has been found to be as thick as 12 feet. A yellow-brown to orange limonitic soil occurs below the hematitic zone. In this zone, concentration of nickel increases and cobalt decreases. An increase in magnesium and a decrease in iron also occurs. The thickness of this zone averages 12 feet but may be as great as 35 feet. Below is found a greenish-white saprolitic-serpentinitic zone which is transitional to the unaltered peridotite. Typically, this zone has no mineralization.

Mining History in the Area

Mining has been part of the area's history since the discovery of gold in 1851. Gold has been mined on a small scale since that time in Smith River gravels and in lode deposits in the Myrtle Creek drainage. Copper and silver were mined in the Altaville area, northwest of Gasquet Mountain, from 1860 to 1880. Chromite was first discovered in the 1850's in the Diamond Creek area, also northwest of Gasquet Mountain, and was periodically mined until 1955. Platinum nuggets have been found with gold nuggets in placer deposits since the late 1800's.


Surface
Dehydration progressive toward surface
Cap of iron oxides
Nodules of iron oxides
Zone of residual laterites
Porous laterites
Limit of separation of nickel with residual iron
Zone of altered peridotite and of the nickel concentration
Zone of fresh peridotite
Migration of nickel downward
Peridotite altered mineralized
Fresh peridotite with tenor 0.25 nickel

From "Ore Deposits," Park and MacDearmid, 1975

FIGURE 5-11

TYPICAL PROFILE THROUGH NICKELIFEROUS LATERITE DEPOSITS

Dames & Moore

86B

Portions of California Nickel's claims were originally held by private parties prior to 1950 and were leased to major miing companies between 1950 and 1968. Between 1976 and 1980, California Nickel acquired various claims which now comprise the proposed mining project and in 1977, California Nickel began a major exploration effort. Exploration activities have included drilling, pit excavation, road building, and reclamation experiments.

Initial plans for processing laterite ores involved a multi-stepammonia-leach process for metal extraction (the UOP process). After further review, California Nickel opted for an acid-leach process, and a pilot plant was constructed at Hazen Research, Golden, in the fall, 1981.

California Nickel's Proposal

The design rate of the process facility is 4000 dry short tons per day. The process facility will operate 330 days a year, and the mine will produce 1.32 million short tons of ore per year. At the estimated annual production rate, mining will disturb about 120 to 150 acres per year. Proven ore reserves cover 12 years of mining, while possible and potential reserves cover another six years (Table 5-3).

Initial mining will be along the crest of Gasquet Mountain, where slopes are moderate and mining and reclamation problems are considered to be least troublesome. Later mining will move onto steeper slopes, but never to slopes steeper than 50%. Only the first three years' mining blocks have been delineated. Other blocks will be identified by a sampling program. Cutoffs for mining blocks are .80% nickel and a 2 feet thickness of laterite.

TABLE 5-3

GASQUET MOUNTAIN PROJECT ORE RESERVES[a]

	Dry short tons	Acres	% Ni	% Co
Proven Ore	16,007,000	1,569.1	0.75	0.071
Probable Ore	346,000	(b)	0.75	0.070
Possible Ore	7,237,000	1,404.4		
TOTAL	23,590,000	2,973.5		

NOTES:

a - Based on a cutoff of 0.8% equivalent nickel
b - Probable ore is located beneath areas of proven ore.

(Kaiser Engineers, 1982)

Mining Method

As noted, only the first three years mining have actually been identified. In any operational year, sampling and surveying will identify the next year's mining area, and the area and road system will be individually approved by the Forest Service each year. Areas with uneconomical ore grades or thickensses and areas containing endangered plants will be left as "greenbelt" areas.

Ore zones will be mined from the surface with a standard backhoe (Figure 5-12). In any mining block the first cut will be made at the lowest topographic part of the block, and further mining will proceed upslope. Trucks will haul the ore to the grizzly station (coarse screen used for rough sizing) at the wash plant. Material larger than five inches will be scrubbed and separately stockpiled for return to the mined-out areas. It will not be processed.

The backfilling process is fundamental to debates about the viability of reclamation proposals. The reclamation proposal is outlined below and illustrated in Figure 5-13:

1. Trucks will first dump coarse rock in the mine cut, which is then spread by bulldozer to form a haul road.
2. Trucks will then haul processed fines (the tailings) to the cut and sequentially dump the material on the downhill side of the haul road.
3. Bulldozers will grade this material, push it against the next downhill berm, and fill the cut until it reaches the berm top.
4. At the next uphill cut, a backhoe forms a new retaining berm for the next filling pass.

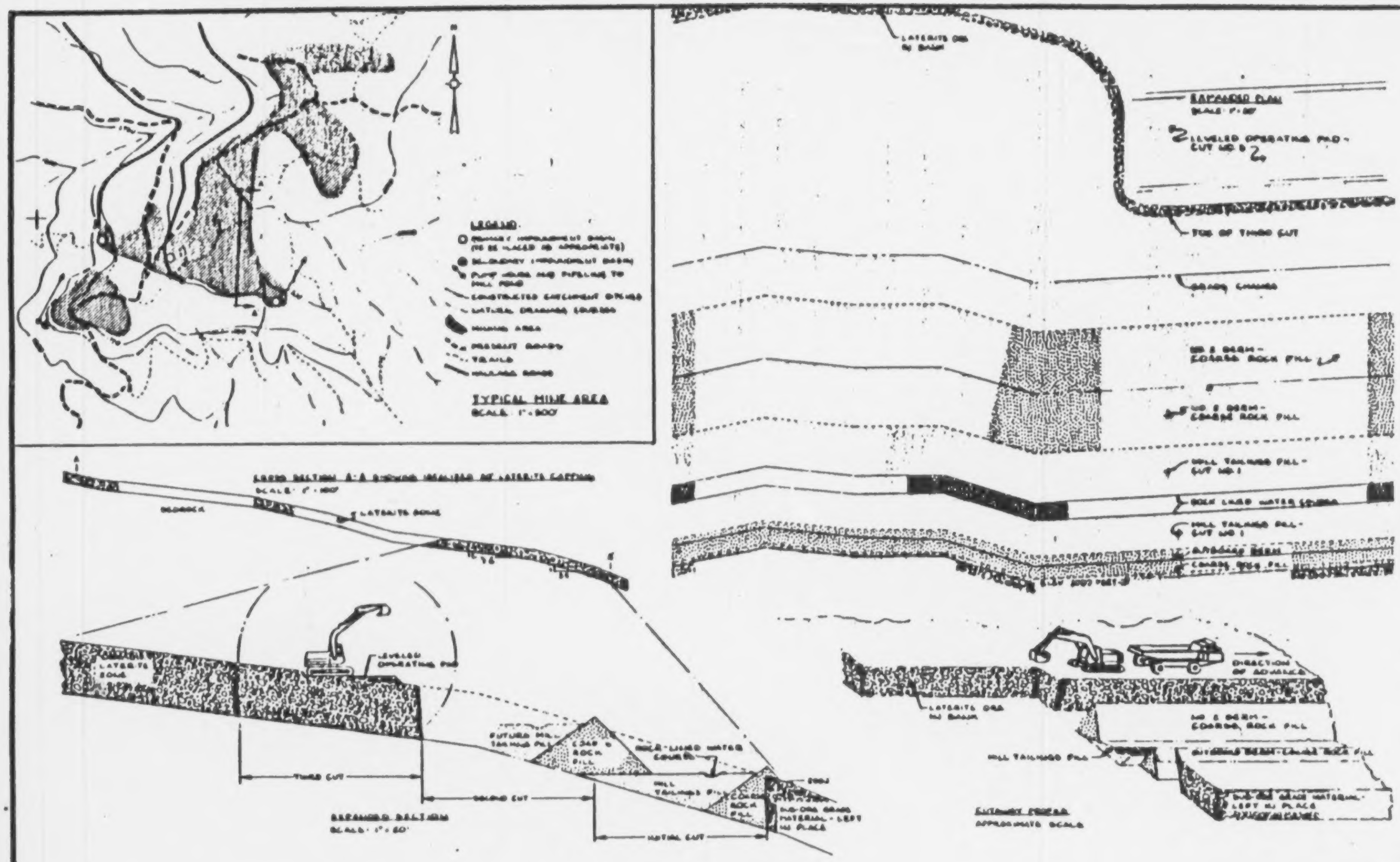

LEGEND
PRIMARY IMPOUNDMENT BASIN (TO BE PLACED AS APPROPRIATE)
SECONDARY IMPOUNDMENT BASIN
PUMP HOUSE AND PIPELINE TO MILL POND
CONSTRUCTED CATCHMENT DITCHES
NATURAL DRAINAGE COURSES
MINING AREA
PRESENT ROADS
TRAILS
HAULAGE ROADS
TYPICAL MINE AREA
SCALE 1"=800'
CROSS SECTION A-A
SCALE 1"=100'
BEDROCK
LATERITE ZONE
LATERITE ZONE
LEVELED OPERATING PAD
FUTURE MILL TAILINGS FILL
COARSE ROCK FILL
ROCK-LINED WATER COURSE
MILL TAILINGS FILL
COARSE ROCK FILL
SUB-ORE GRADE MATERIAL - LEFT IN PLACE
THIRD CUT
SECOND CUT
INITIAL CUT
EXPANDED SECTION
SCALE 1"=50'
LATERITE ORE IN BANK
EXPANDED PLAN
LEVELED OPERATING PAD - CUT NO. 2
TOE OF THIRD CUT
GRADE CHANGE
NO. 3 BERM - COARSE ROCK FILL
NO. 2 BERM - COARSE ROCK FILL
MILL TAILINGS FILL - CUT NO. 1
ROCK LINED WATER COURSE
MILL TAILINGS FILL - CUT NO. 1
COARSE ROCK FILL
DIRECTION OF ADVANCE
LATERITE ORE IN BANK
NO. 3 BERM - COARSE ROCK FILL
MILL TAILINGS FILL
SUB-ORE GRADE MATERIAL LEFT IN PLACE
CUTAWAY PROFILE
APPROXIMATE SCALE

Figure 5-12

MINING SEQUENCE

Source: Kaiser Engineers

Dames & Moore

G 38

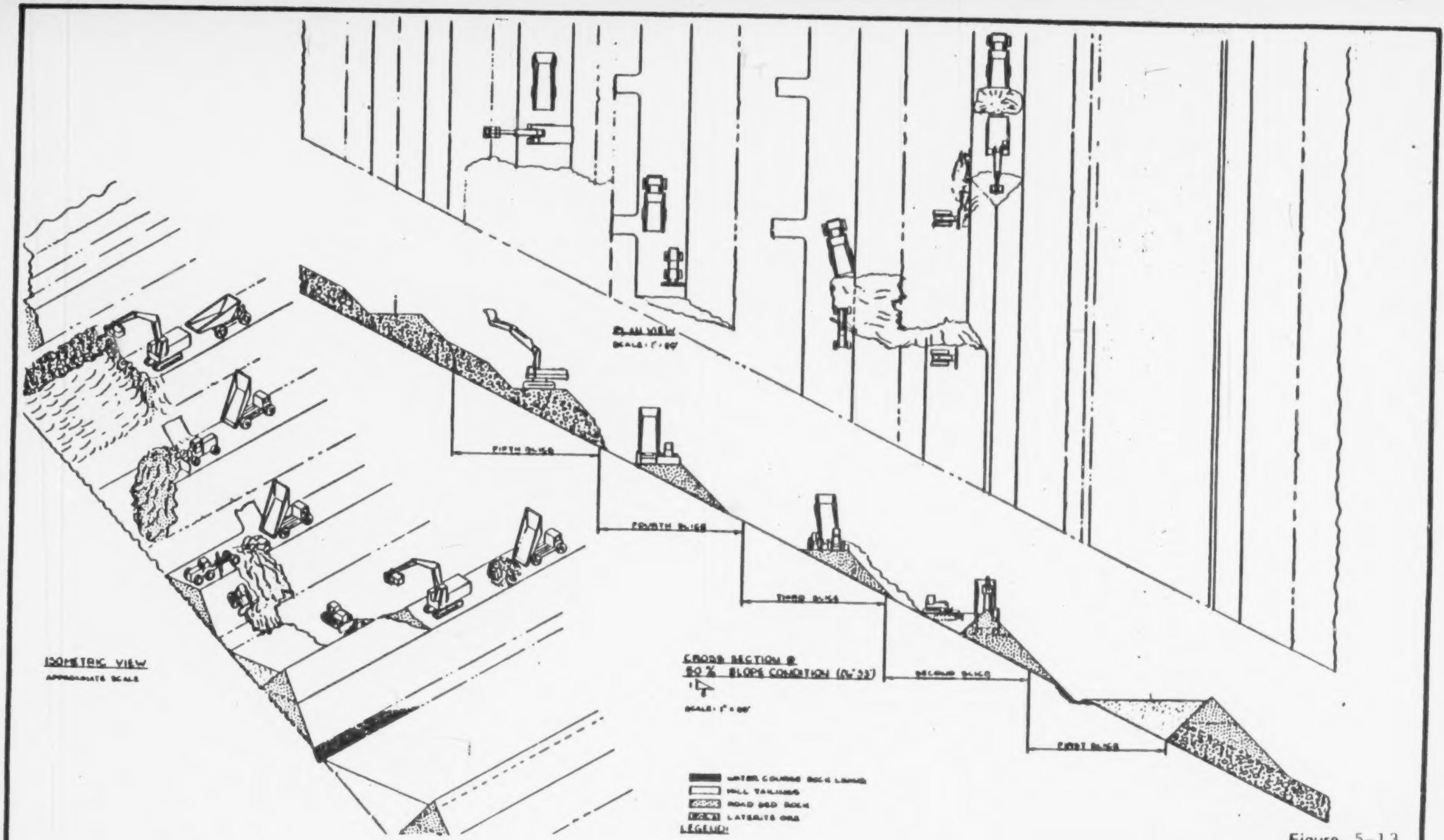

PLAN VIEW
SCALE: 1" = 80'
FIFTH SLICE
FOURTH SLICE
THIRD SLICE
SECOND SLICE
FIRST SLICE
CROSS SECTION @
50 % SLOPE CONDITION (26°33')
1
2
SCALE: 1" = 80'
ISOMETRIC VIEW
APPROXIMATE SCALE
WATER COURSE ROCK LINING
MILL TAILINGS
ROAD BED ROCK
LATERITE ORE
LEGEND:

Figure 5-13

BACKFILLING SEQUENCE

Source: Kaiser Engineers

Dames & Moore

All but the upper two feet of tailings backfilled into a cut will be material directly from the processing plant. The upper two feet of tailings will actually be a mixture of coarse and fine tailings, native mulch produced from land clearing, furnace ash produced by the coal-fired power plant needed for the project, and fertilizer. This mixture will have been blended at the processing plant and no discing will be required at the mine cuts. The area will be subsequently planted with grasses and forbs, and seedling trees and shrubs will later be planted. Refertilization of areas will periodically be necessary, and replanting is proposed for any areas where less than 10% cover develops after one year.

Drainage control is proposed with an extensive system of ditches, settling ponds, and outlets to control winter runoff.

Part of the proposal is construction of a dam across Copper Hill Creek, a tributary of Hardscrabble Creek. The dam will meet water supply needs of the process facility, which previously had been expected to be fulfilled by wells. The dam would be 230 feet high, as tall as a 15-story building, and would have a maximum capacity of 957 acre-feet. It would be the first dam constructed in the entire Smith River drainage basin, although Copper Hill Creek is not a perennial stream.

Processing Facilities

After the coarse (greater than 5 inch diameter) material is screened, and the oversize material returned to the mine, the remaining material will be subjected to attrition in order to remove the soft, clay-like fine ore from the larger particles. The larger particles contain somewhat less nickel and cobalt. The remaining ore fractions are then separated into those containing subsstantial amounts of chromite ($FeCr_2O_4$), magnetite (Fe_3O_4), and magnesium. The chromite and magnetite are recovered by methods that utilize the differences in physical properties of the different minerals.

The higher value nickel- and cobalt-bearing ore is leached, using sulfuric acid under various temperature and pressure combinations. Most of the ore is treated at high temperatures and pressures in order to obtain complete solubilization of nickel, cobalt, and magnesium. Ore with higher concentrations of magnesium can be processed at lower temperatures and pressures if greater magnesium recoveries are desired.

In order to produce high-grade products that are solubilized during the leach process, deleterious elements, such as iron, aluminum, chromium, manganese, zinc, and copper, must be eliminated. The purification steps yield one solution containing nickel and cobalt and another consisting principally of magnesium sulfate. The nickel and cobalt are recovered in separate solutions, using liquid ion exchange. The entire process is a multiple step, complex process.

Regulatory, Reclamation, and Environmental Issues Related to the Proposed Project

The Gasquet Mountain project is controversial. Although many citizens support the project, several groups in and outside the immediate area have voiced concern about the impacts of the project. The North Coast Environmental Center of Arcata and Cal-Nickel's Neighbors of Del Norte County have prepared analyses of adverse impacts of the project, and other organizations firmly oppose it. The Sierra Club published an unfavorable article on the project in its national magazine (Berthelsen, 1982). Numerous state and federal agencies have reviewed different aspects of the California Nickel proposal. Particular concern about the project, and critical comments, have been offered by the California Air Resources Board, Del Norte County, and the California Surface Mining and Reclamation office. The delay in publication of the draft environmental impact statement by the Forest Service and Del Norte County resulted from continuing analysis and data gathering that the proposal necessitated.

Major concerns about the project include the following:

1. The ability to revegetate tailings and the effect of uncontrolled erosion on the high quality waters of the Smith River basin;
2. The impact of the proposed dam on Hardscrabble Creek and the precedent it sets for dam construction in the entire Smith drainage basin;
3. The air quality effects of the proposed power plant and processing facility on surrounding forest, especially Redwood National Park;
4. The effect of the project on the endangered plant communities of Gasquet Mountain; and
5. The socioeconomic impacts of the project and creation of a bust cycle for the region at the close of mining.

Each of these issues is a substantive one, and the detail of analysis in the environmental impact statement will be a critical factor in determining the nature of future debate on the project and whether or not litigation on the project ensues.

Revegetation of tailings is a critical issue affecting viability of the project. As noted in the previous section, California Nickel is mining the soil and subsoil because these horizons are the ore. Thus, unlike mining in other parts of the United States, there will be almost no salvaging of topsoil for future use in reclamation. California Nickel proposes to refill mine pits with processed tailings, and the upper two feet of tailings will be a mixture of coarse and fine tailings, power plant fly ash, fertilizer, and mulch. The mixture of these materials is based on laboratory experiments, primarily carried out at the University of California, Berkeley, in finding species that will grow in the processed tailings. The tailings have been identified as deficient in nutrients, which limits revegetation success.

As has been noted, the existing soils are not very productive and do not support dense stands of vegetation. However, these soils are also not very erosive, because of their high permeability. Rainfall sinks into the soil, rather than running over the surface and eroding the soil. Under reclamation conditions, much finer grained and thus less permeable material will be replaced in the mine pits. Even if the same amount of vegetation cover were achieved after reclamation as before mining occurred, more erosion would occur following reclamation due to the changed soil conditions.

Citizens opposing the proposed project have objected to the limited extent of laboratory revegetation studies which they believe to be inadequate to prove reclamation success at Gasquet Mountain. The Forest Service is taking the position that these greenhouse experiments are adequate. Obviously, if revegetation were to fail at the site, under the heavy rainfall conditions of the area, erosion would increase, possibly to the detriment of the Smith River system. In a draft supplied to Earth Resource Associates of the now published EIS, the Forest Service (1983) only considered a worst-case scenario, which considered revegetation failure on one year's mining area (about 150 acres). Citizens may insist on a scenario that examines failure of a greater amount of the proposed 3000 acre mining area. Such a large-scale worst-case scenario is customarily evaluated in many impact statements.

Conversations with the California Air Resources Board (Sassenrath, 1982) indicate that air quality concerns may be mitigable. Environmentalists contend that sulfur and nitrogen oxide emissions from the California Nickel facility complex will adversely affect surrounding forests, particularly those of Redwood National Park. These concerns have been the basis of continuing dialogue between regulatory personnel and the company. California Nickel has made some changes in its proposal to accommodate these concerns. Sassenrath (1982) feels that standard scrubber technologies can mitigate emission problems with the plants.

The most important delay in the air quality review process, according to Sassenrath, has been California Nickel's failure to file appropriate applications with the Air Resources Agency. When California Nickel filed its first environmental impact statement as well as its PSD permit application, the company had never contacted the state air agency (Sassenrath, 1982). The application was found to be incomplete because the company did not address several issues--such as a determination of necessary control technologies--that are required in an application. At present, the Forest Service EIS is being delayed while the company submits additional data on air quality issues. However, if the proper regulatory channels are followed, the preliminary feeling of the state staff is that Class I and II standards can be met by the facilities. There is no indication as to whether environmentalists will agree.

The Hardscrabble Dam is of concern to environmentalists because it would be the first dam in the Smith River basin. They are concerned that this dam will serve as a precedent for further dam construction in the basin for interbasin transfers to southern California. The issue of continued transfer of northern California water to the south is of great concern to northern Californians.

Review of an intitial draft of the Forest Service EIS supplied to Earth Resource Associates indicates that much of the proposed action described in the document is conceptual. No specific designs are proposed for drainage ditches or other water control facilities, a complete mining plan is not available, and other details are lacking. Thus, the impact analysis cannot be as specific as it could if details were available. In light of the controversy surrounding the project, the conceptual nature of many of the proposals may prove to be a point of contention with environmentalists.

Other Concerns

As noted earlier, California Nickel has changed its proposed processing technology to an acid leach system. The grade of ore cutoff at the mine is proposed to be 0.8% nickel, but may actually be lower. Some metallurgists have been skeptical about this process for the following reasons:

1. The proposed process is less reliable and less tested than other processes for laterite ores;

2. The cutoff grade at California Nickel is well below ore grade cutoffs at other mines;

3. The economics of the Gasquet project depend on adequate returns for three products; and if threshold prices are not achieved for each, the project may not be economically viable.

In a general sense, the Gasquet project necessitates the use of a less documented recovery process using poorer ores than used anywhere else in the world, and the economics of the project demand price supports for nickel, cobalt, and chromium. A more detailed comparative analysis of hydrome Halurgical processing is presented in Chapter VI. In light of the serious environmental opposition to the project, the serious questions about reclamation, and the conceptual nature of some of the mine planning, the short-term viability of this project is questionable.

STILLWATER COMPLEX/BEARTOOTH FACE

The Stillwater Complex is a group of layered igneous rocks, exposed as a rough lens 28 miles long by one to five miles wide along the northern margin of the Beartooth Mountains in south-central Montana. The Complex "contains the largest nickel resources in the United States" (Page and Dohrenwend, 1973, p. 1). Chromite was briefly produced from the Complex during World War II and from 1953 to 1961, in both instances under government contract. In 1981, the Anaconda Minerals Company filed an application with federal and state agencies to develop a platinum-palladium mine and mill in the area, and a draft environmental impact statement on the project was issued in June, 1982. PGM Resources, a joint venture of the Manville Corporation and Chevron, has also proposed a mine in the complex. Economic uncertainties have caused both projects to be at least temporarily delayed. PGM Resources reportedly is considering acquisition or a joint venture with the Anaconda Minerals project. There are no current proposals to mine chromium or nickel in the complex.

Environmental Setting

The Beartooth Mountains are a nearly level, treeless series of high plateaus dotted with lakes of various sizes and strewn with the rocky remains of glaciers. In a central part of the area, the spine of mountains rises to a series of alpine peaks, which include the highestpoint in Montana, Granite Peak (12,799 ft). Much of the plateau surfaces are above 9000 ft and are characterized by alpine tundra vegetation. Plateaus are dissectd by steep-walled drainages that were scoured by Pleistocene-aged glaciers. The Stillwater Complex is exposed along the northern edge of the Beartooth Mountains in an area known as the Beartooth face.

Along the Beartooth face, the plateaus drop steeply to the surrounding foothills, where elevations are generally less than 5000 feet. The dominant land use of the foothills is cattle ranching (grazing and hay production) with some second home development. The vegetation is grasslands (bluebunch wheatgrass, Idaho fescue, needlegrasses) or sagebrush shrublands, with aspens and chokecherries found in ravines. The high slopes of the Beartooth face are in subalpine fir (7000-9000 feet), Douglas fir (5000-9000 feet), or lodgepole pine (6000-9000 feet) forest.

The climate of the area is mountainous continental and is characterized by large diurnal and annual temperature ranges and marked differences in precipitation, temperature, and wind patterns over distances of only a few miles. Air quality in the area is excellent.

Streams of the northern Beartooth Mountains drain to the Yellowstone River and are of good to excellent quality. The trout fishery of these streams is of statewide importance, according to the Montana Department of

Fish, Wildlife, and Parks. Virtually all of the streamflow in the area is derived from snowmelt and runoff from high elevations.

A diverse wildlife resource exists, including elk, mule deer, moose, bighorn sheep, cougar, and black bear. The bald eagle inhabits the area at least seasonally, and unconfirmed sightings or signs have been reported of the Northern Rocky Mountain wolf, peregrine falcon, and grizzly bear.

The amenities of the area make it a focus of recreational use. The high country of the Beartooth Plateau is included in the 900,000 acre Absaroka-Beartooth Wilderness. The northern boundary of the Wilderness is near, but does not include, the Stillwater Complex mineral resources. Potential resources of the complex were specifically excluded when the wilderness was established in 1979.

Sociology

The counties in which the Stillwater Complex is located (Park, Stillwater, Sweetgrass) are rural in character. Ranching and farming is the principal industry of the area and has been since the area was first settled. Generally, ranches and farms each average less than 2000 acres, with approximately 250 head of cattle. Many ranches have diversified and also grow crops to help get through periods of depressed cattle markets. During hard times, owners of smaller ranches may be forced to work for larger operations, find employment in town, in Billings (two to three hours' drive to the east), or do contract work for local mining operations. Generally, young people have a difficult time finding employment. If they do not have access to a family ranch or a family-run business, they are forced to choose between the high amenity with lesser employment potential of the local area or higher employment possibility with lower outdoor amenity attractions of larger cities (U.S. Forest Service, 1978).

Many natives of Sweetgrass and Stillwater Counties are descendants of Norwegian farmers and settlers. The U.S. Forest Service (1978) made these observations about the population:

> The townspeople and ranchers have a strong dependence on the land, and they fear a change in the quality of their way of life. They measure this quality by lots of space, little traffic and the other manifestations of masses of people, rural ties, good hunting and fishing, access to the mountains, a community spirit grown from locals' relation to the land and independence (p. 11).

Each of the larger communities located within the study area is characterized by a significant population of retired persons. Many of these people owned or were employed by ranches located throughout the area. Others have come to the area specifically for retirement because of the amenity values of the area.

Geology

The Stillwater Complex is a group of layered igneous rocks which intrudes older Precambrian rocks (to the south) and is unconformably overlain by Middle Cambrian sedimentary strata (to the north). The ore minerals and enclosing rocks formed in horizontal layers as a cumulate or "magmatic sediment" during fractionation of a basaltic magma. The ore zones of primary importance--nickel, chrome, and the platinum group--are found in different parts of this layered assemblage. During formation of the Beartooth Mountains, the Complex was turned on edge, thrust north along faults, and exposed by erosion. In general, layered rocks dip steeply to the north.

For classification purposes, the Stillwater Complex is divided into four stratigraphic zones: the Upper, Banded, Ultramafic, and Basal zones (Figure 5-14). Understanding the mineralogy of the Complex is made relatively simple by the rhythmic occurrence of six minerals: olivine, two orthopyroxenes, augite, chromite, and calcic plagioclase. Chromite, an accessory mineral, is concentrated in 13 alphabetized chromitite layers within the Ultramafic zone. Platinum-group minerals are present in all zones, but most importantly in a sulfide horizon within the Banded zone. Copper and nickel are concentrated in and adjacent to the Basal zone and are present in trace amounts throughout the Complex (Read, 1977).

History of Mining Development in the Stillwater Complex

Prospecting interest in the Stillwater Complex dates to 1883. As was observed throughout the west during this era, a town, Nye, sprang to life around the ensuing mine development, reaching a population peak of 400 in 1887. A small smelter processed gold, silver, and copper, and over 8000 feet of tunnels were driven. The entire mining development collapsed the next year when it was determined that the town, mine, and smelter were located on the then Crow Indian Reservation.

Prospecting and claim staking continued after the Crow Tribe ceded the area in 1890, but there was little success from these explorations. Principal interest in the area was for gold and silver, although the presence of chromium had been known since the first prospecting days. It was not until World War I that some of the chromium deposits of the complex were explored, but no mining took place. After the war exploration ceased, except at the Benbow mine, which was prepared for production in 1929. Financial uncertainties in that year killed the project.

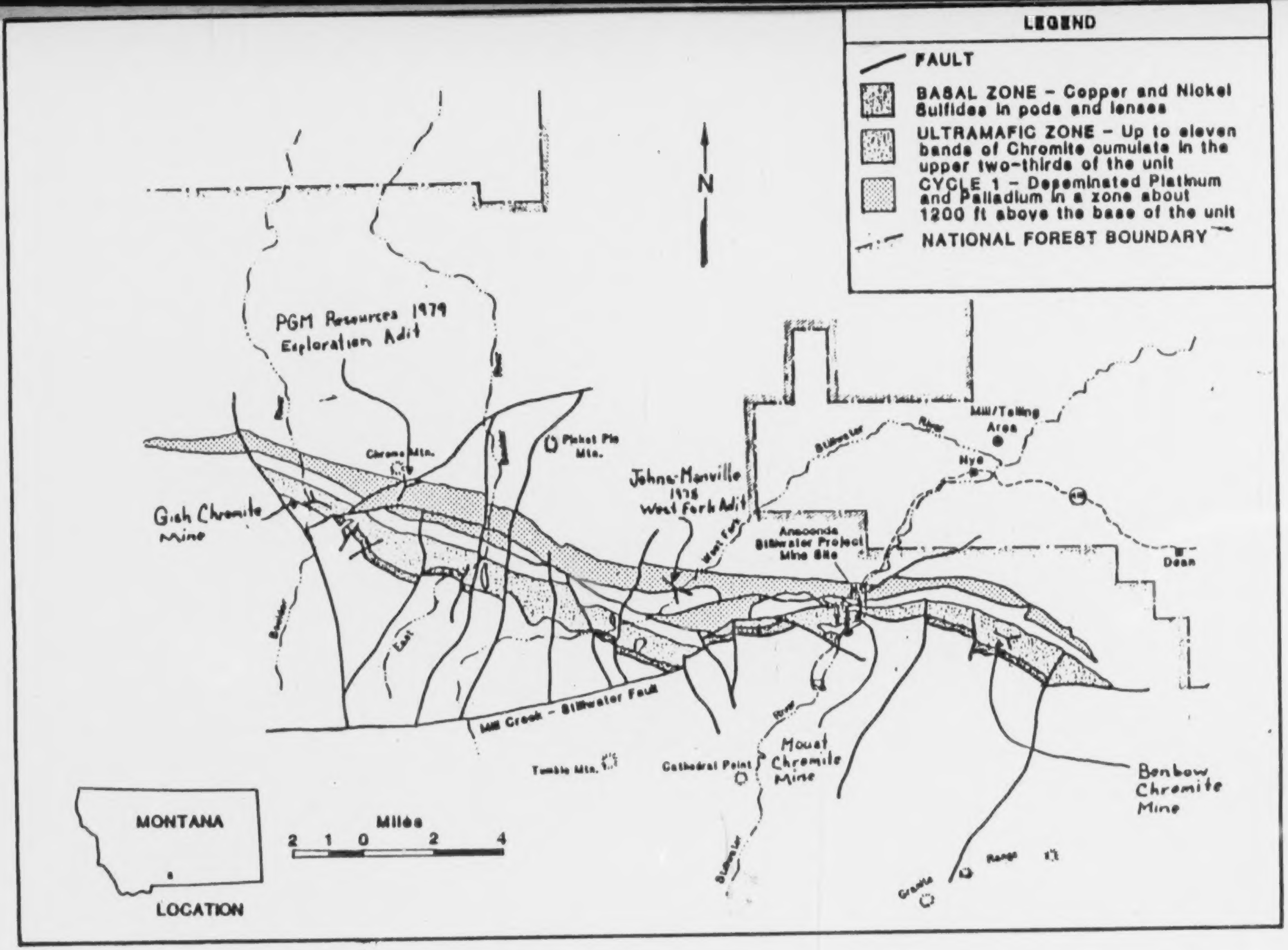

LEGEND
FAULT
BASAL ZONE - Copper and Nickel Sulfides in pods and lenses
ULTRAMAFIC ZONE - Up to eleven bands of Chromite cumulate in the upper two-thirds of the unit
CYCLE 1 - Deseminated Platinum and Palladium in a zone about 1200 ft above the base of the unit
NATIONAL FOREST BOUNDARY
N
PGM Resources 1979 Exploration Adit
Chrome Mtn.
Picket Pin Mtn.
Johns-Manville 1978 West Fork Adit
West Fork
Stillwater River
Mill/Tailing Area
Nye
Dean
Anaconda Stillwater Project Mine Site
Gish Chromite Mine
Boulder
East
Mill Creek - Stillwater Fault
Tumble Mtn.
Cathedral Point
Mouat Chromite Mine
Benbow Chromite Mine
Stillwater
Granite
Range
MONTANA
LOCATION
Miles
2 1 0 2 4

FIGURE 5-14 --Generalized Geologic Map of the Stillwater Complex

World War II brought new interest in the area. The U.S. Bureau of Mines began studies in the area in 1939. As a result of threatened disruption of foreign chromite imports, the Anaconda Copper Mining Company, as agent for the Federal Metals Reserve Corporation, began developing the Benbow property in June, 1941. Soon afterward, development work was expanded to include the Mouat-Sampson property and in spring, 1942, the Gish property (Figure 5-14). Production was 102,000 tons assaying at 40% Cr_2O_3 using ore averaging 20% Cr_2O_3. More than $12 million was spent by the federal government on camps, plants, equipment, development, and operation in the Complex (Wimmler, 1948).

Actual production at the Benbow and Mouat mines ended in mid-1943 when high-quality imported ore again bacame available. Government leases on the properties in the Complex were cancelled, and numerous buildings and other equipment were dismantled and sold.

In 1953, the government subsidized contracts of the American Chrome Company, a subsidiary of Goldfield Consolidated Mines, which "effectively reduced costs during fulfillment of its contract but had to discontinue operations because the chromium to iron ratio (of the concentrate) could not be improved to meet industry requirements (Read, 1977, p. 4). Cost of this operation to the U.S. government was $30.00 per ton of Cr_2O_3. Mining was undertaken by American Chrome using long hole and shrinkage stoping with geologic conditions dictating which technique would be used. Long-holing was used for veins under 6 feet (Huttl, 1954).

The 980,000 ton stockpile (36% to 38% Cr203; Cr/Fe ratio = 1.5) created by this mining, which continued until 1961, was sold to Metallurg, Inc. in 1974 for $9.00 per ton of CO_2O_3. The chromium stockpile has thereafter been gradually removed and less than half of the original stockpile remained in 1982.

Since the late 1960's, major mineral-related activities in the Complex have involved exploration for copper-nickel sulfide ores and platinum group minerals. The copper-nickel deposits of the Complex are important for their nickel. Three companies own the biggest blocks of these deposits--Anaconda Minerals, AMAX, and Cypres Mines. No development, however, has been proposed for these deposits.

Johns-Manville Sales Corporation, which initiated exploration in the area in 1967, identified four zones of high platinum-group metal concentrations. One of these zones was found to be nearly continuous for about 25 miles, and the company filed claims in a large part of the Stillwater Complex (Figure 5-15). Johns-Manville initiated underground exploration in the north wall of the West Fork Stillwater River valley. The company encountered ore grades averaging 14 parts per million platinum, two to four times those typical of the Merensky Reef in South Africa. However, the company was forced to abandon this adit in 1976 when water under very high pressure was encountered (Montana Department of State Lands and U.S. Forest Service, 1982). Johns-Manville was fined $10,000.00 by the Montana Department of Health and Environmental Sciences for uncontrolled discharges of the adit waters to the West Fork. The only contaminants to the river, however, were ammonia and nitrate from the blasting materials (Western, 1979). In 1979, Johns-Manville, in partnership with Chevron USA began development of a second exploration adit (the Frog Pond) further west in the Complex.

Anaconda owned the mineral rights to the area around the then abandoned Mouat chromite mine, and in 1977 the company identified a zone of platinum-palladium mineralization on its property. In mid-1979 the company began construction of the Minneapolis adit and reached the mineralized zone in mid-

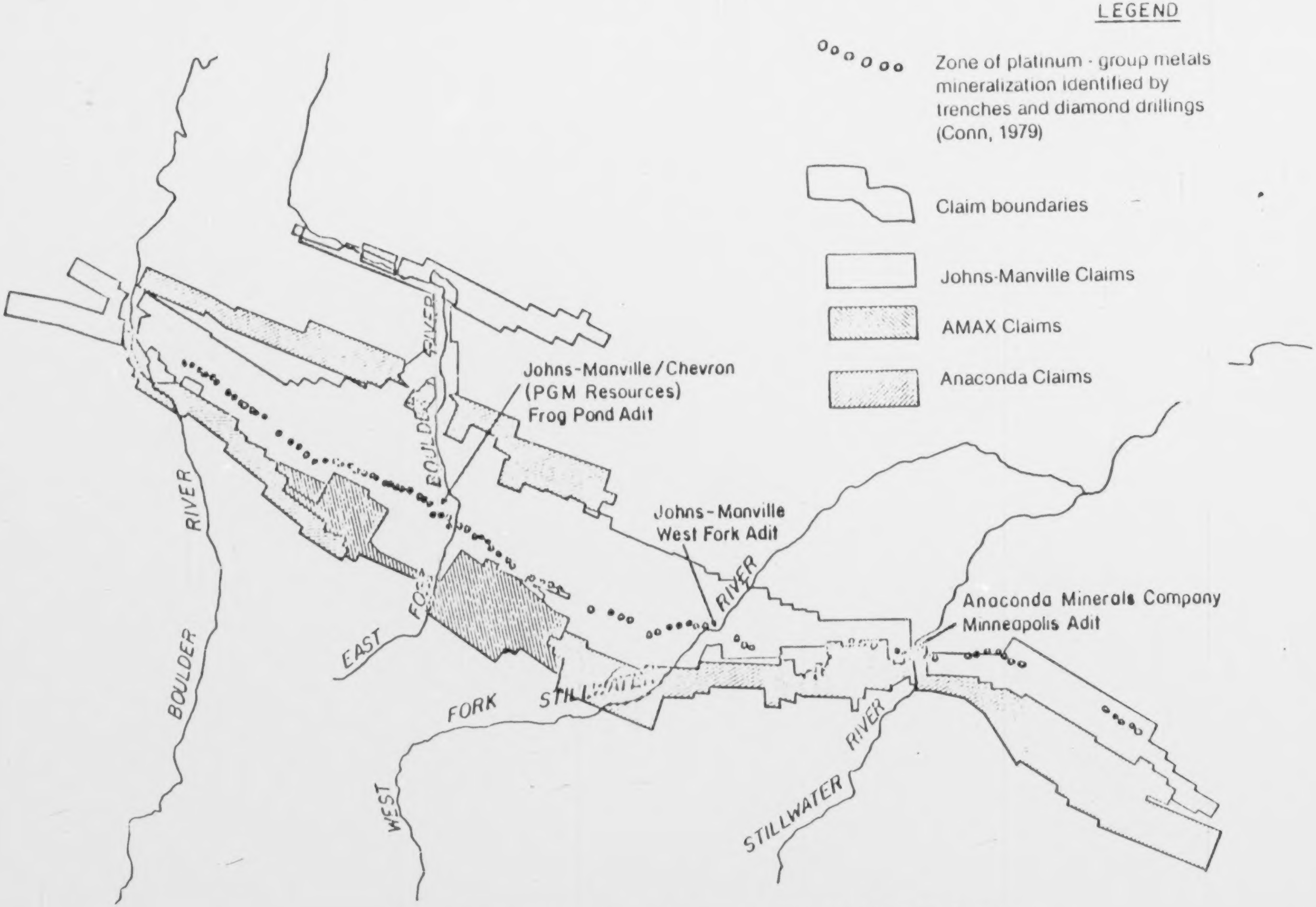

LEGEND
Zone of platinum - group metals mineralization identified by trenches and diamond drillings (Conn, 1979)
Claim boundaries
Johns-Manville Claims
AMAX Claims
Anaconda Claims
Johns-Manville/Chevron (PGM Resources) Frog Pond Adit
Johns-Manville West Fork Adit
Anaconda Minerals Company Minneapolis Adit
BOULDER RIVER
BOULDER RIVER
EAST FORK
WEST FORK STILLWATER RIVER
STILLWATER RIVER

FIGURE 5-15 --Mineral Ownership of the Stillwater Complex

December. Since issuance of its exploration permit, drifts and raises have been developed and bulk samples obtained and analyzed.

Exploration and development activities have continued on both projects; and as noted above, Anaconda Minerals formally applied for a mining permit in August, 1981, for a 1000 tpd operation. Stillwater PGM Resoruces (the name taken by the Johns-Manville/Chevron partnership) began environmental baseline studies in 1981 and had anticipated permit application submittal in 1982 for a 2500 tpd operation. Declining economic conditions have now delayed both projects. No drilling took place at either site in 1982 and equipment has been hauled away from the Anaconda site. Anaconda Minerals is now seeking a partner for its project, and PGM Resources reportedly has first option to buy the Anaconda property (Northern Plains Resource Council, 1982). Stillwater PGM Resources "put on hold" its plan to construct roads to new low-level adit in the western part of the Complex, and at least temporarily has discontinued operations in the Boulder River part of the Complex. Instead, PGM Resources concentrated its efforts in 1982 on evaluation of the Anaconda project. There are several bases for PGM Resources' interest in Anaconda Minerals' holdings:

> Advantages to extracting the ore from the Stillwater side of the Complex includes the utilization of existing facilities, an adit site, tailings pond, and already constructed roads. Also, Anaconda is farther along with the preparation of their EIS statement . . . no decisions on throwing in with Anaconda have been made and it is just one of a number of alternatives PGM is exploring. Another possibility is that (PGM) would buy out Anaconda.
>
> PGM is presently looking at some of Anaconda's claims which lie adjacent to PGM's claims. Further exploration and negotiations will take place before action, such as offering to buy the property or forming a partnership, is contemplated (Big Timber Pioneer, June 9, 1982).

No timetable is, therefore, available for a formal permit application in the Boulder River portion of the Complex. Speculation holds that in the event of a "one-mine scenario), rather than two mines, ultimate operational size would be 2500 tpd.

ANACONDA MINERALS - STILLWATER PROJECT PROPOSAL

Figure 5-16 shows the topography of the Anaconda permit area and Figure 5-17 indicates the location of the project area in the Stillwater River Valley. A summary of the proposed mining and milling method has been included in the project EIS:

> The proposed mining method is shrinkage stoping, a method used in steeply dipping mineral veins where the surrounding rocks provide strong walls. The ore would be mined in successive flat or inclined slices working upward from underground passages. After each slice is blasted down, enough broken ore would be removed from below to allow working space between the top of the pile of broken ore and the bottom of the unbroken ore zone (Figure 5-18).
>
> Underground excavations, or stopes, would be developed in pairs, each 195 ft high by 397 ft long. Five pairs of stopes would be worked at one time. At each end of a "double stope" a passageway would be advanced to provide access, ventilation, and services.
>
> Vertical support pillars 40 ft long would be left between stope pairs and a supportive (crown) pillar would be left between the uppermost mining horizon and the surface. Passageways would be horizontally excavated at 197-ft vertical intervals. Through these passageways the ore and waste rock would be hauled via the Minneapolis adit to the surface. Mining would proceed from top to bottom and generally from west to east. The development phase of the operation would occur over a period of approximately 30 months, including about 15 months for shaft development. Production of ore is expected to average 1000 tons per day and peak at 1200 tons per day . . . Ore would be transported 24 hours a day, seven days a week to the mill in 20- to 25-ton highway trucks on county road FAS (Federal Aid Secondary) 419 and 420 and would run at approxiamtely 30-minute intervals. It is estimated that 50-60 loaded trucks and 50 to 60 empty return trucks would use FAS 419 and 420 from the mine to the mill daily . . .
>
> Three surface openings would be utilized for the Stillwater Project: the Minneapolis adit (5180 ft of elevation), the 1980-level adit (6530 ft of elevation) and a powder magazine . . . (The 1980-level adit is so called because it is at 1980 m of elevation). The two adits are now in place on national forestland, constructed under approved exploration operating plans and permits submitted in 1979-1981 (See Figure 5-15). The Minneapolis adit (9 ft x 11 ft) would be used as the main working entrance to the mine and would allow access for miners, equipment, and supplies. Ore and waste rock would be transported to nearby Anaconda Minerals Company land from this opening. The 1980 level (11 ft x 11 ft) would provide an emergency escapeway, an airway for exhaust ventilation, and a water supply line. The powder magazine would be used for storage and transport of explosives. It would consist of a tunnel and room excavated in the rock face beside the Minneapolis adit . . . .
>
> The mine (is proposed to) . . . operate on a 24-hour-per-day schedule

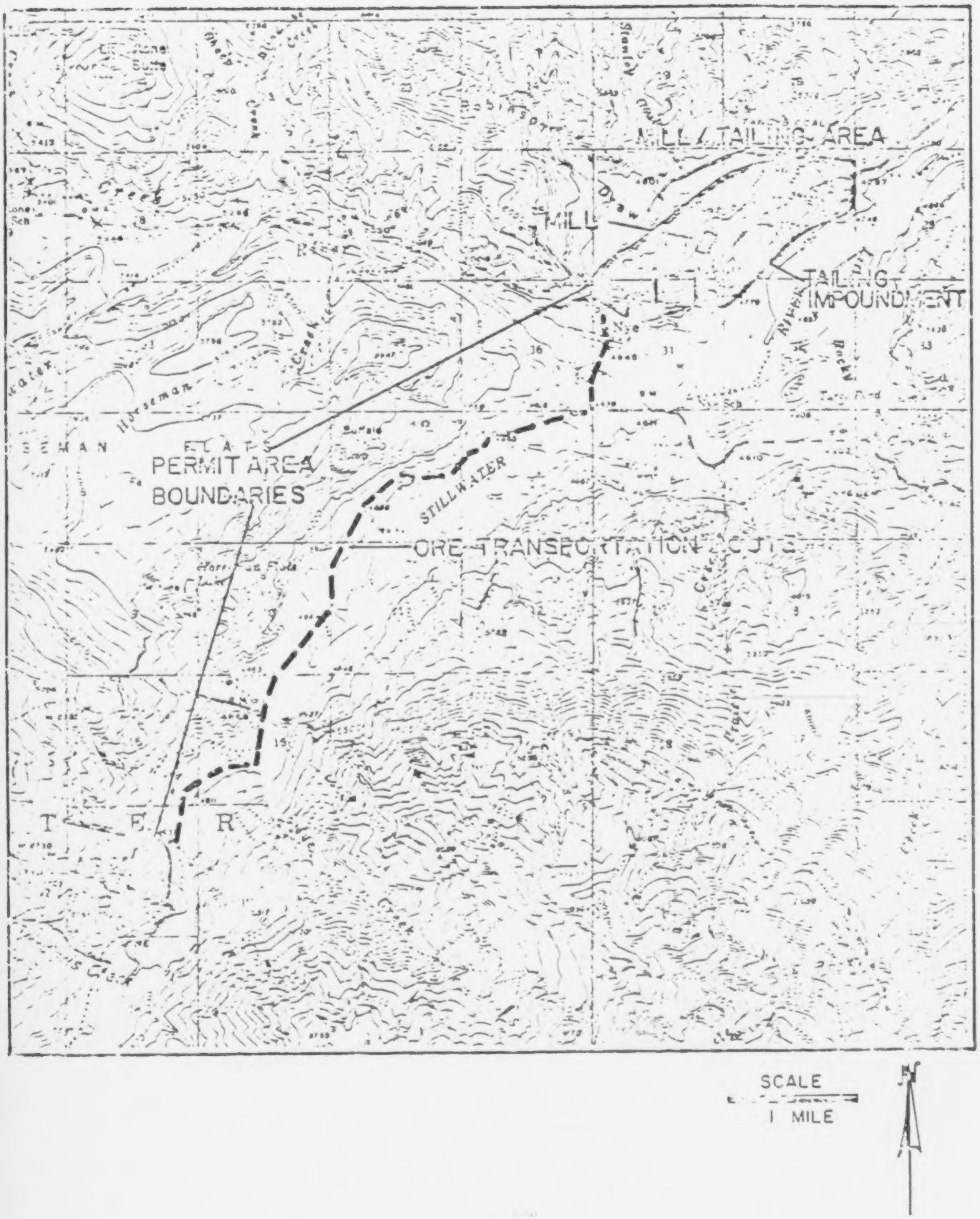

MILL / TAILING AREA
MILL
TAILING IMPOUNDMENT
PERMIT AREA BOUNDARIES
ORE TRANSPORTATION ROUTE
STILLWATER
Horseman
FLATS
Nye
T E R
SCALE
1 MILE

FIGURE 5-16 --Topography of the Permit Area

1030

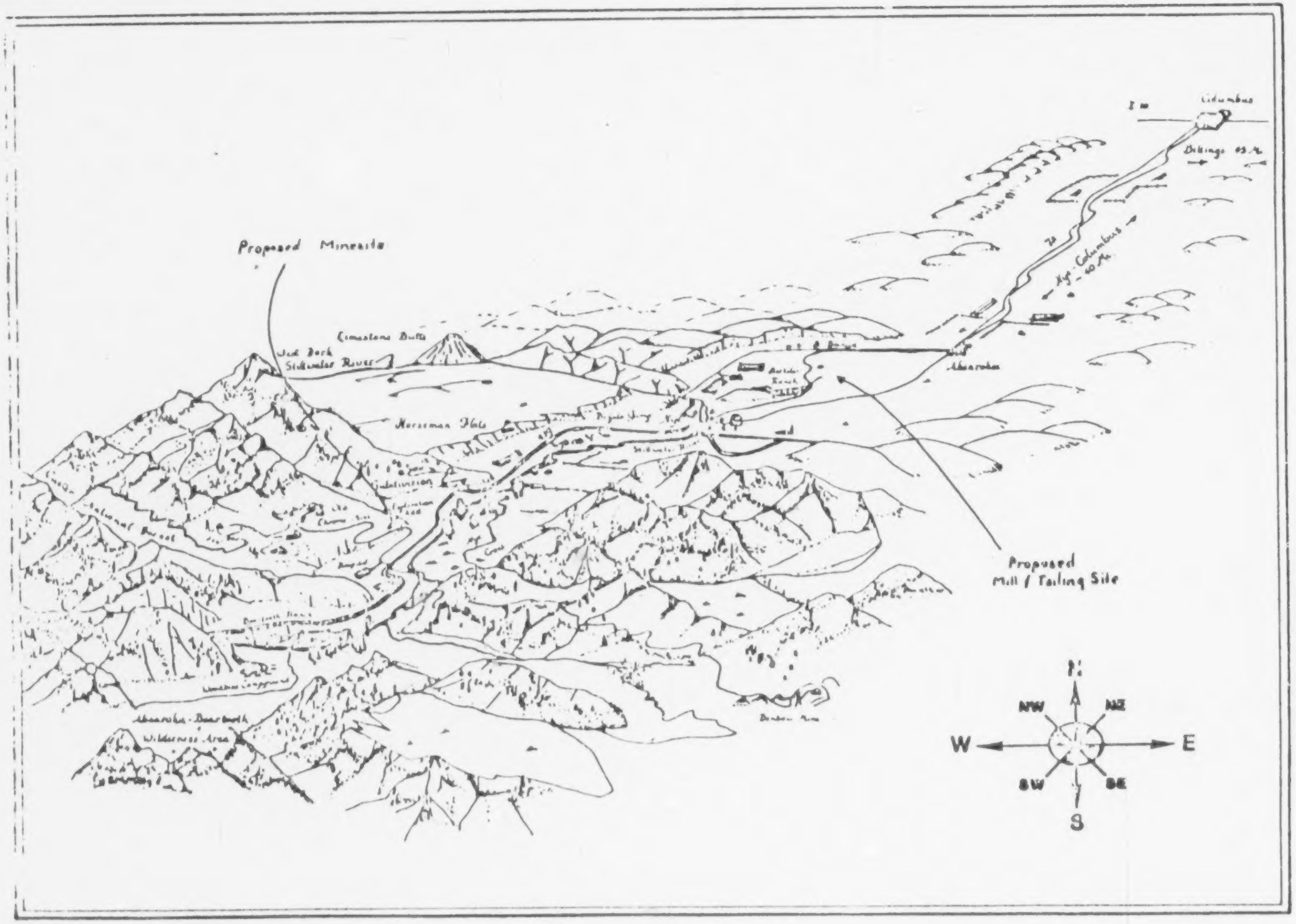

Proposed Minesite
Limestone Butte
West Fork
Stillwater River
Horseman Flats
Stillwater River
Subdivision
Absaroka-Beartooth
Wilderness Area
Absarokee
Columbus
Billings 43 Mi
Hy-Columbus - 40 Mi
Proposed
Mill / Tailing Site
N
NW
NE
W
E
SW
SE
S

FIGURE 5-17-Artists' View of Stillwater River Valley, Showing Project Area

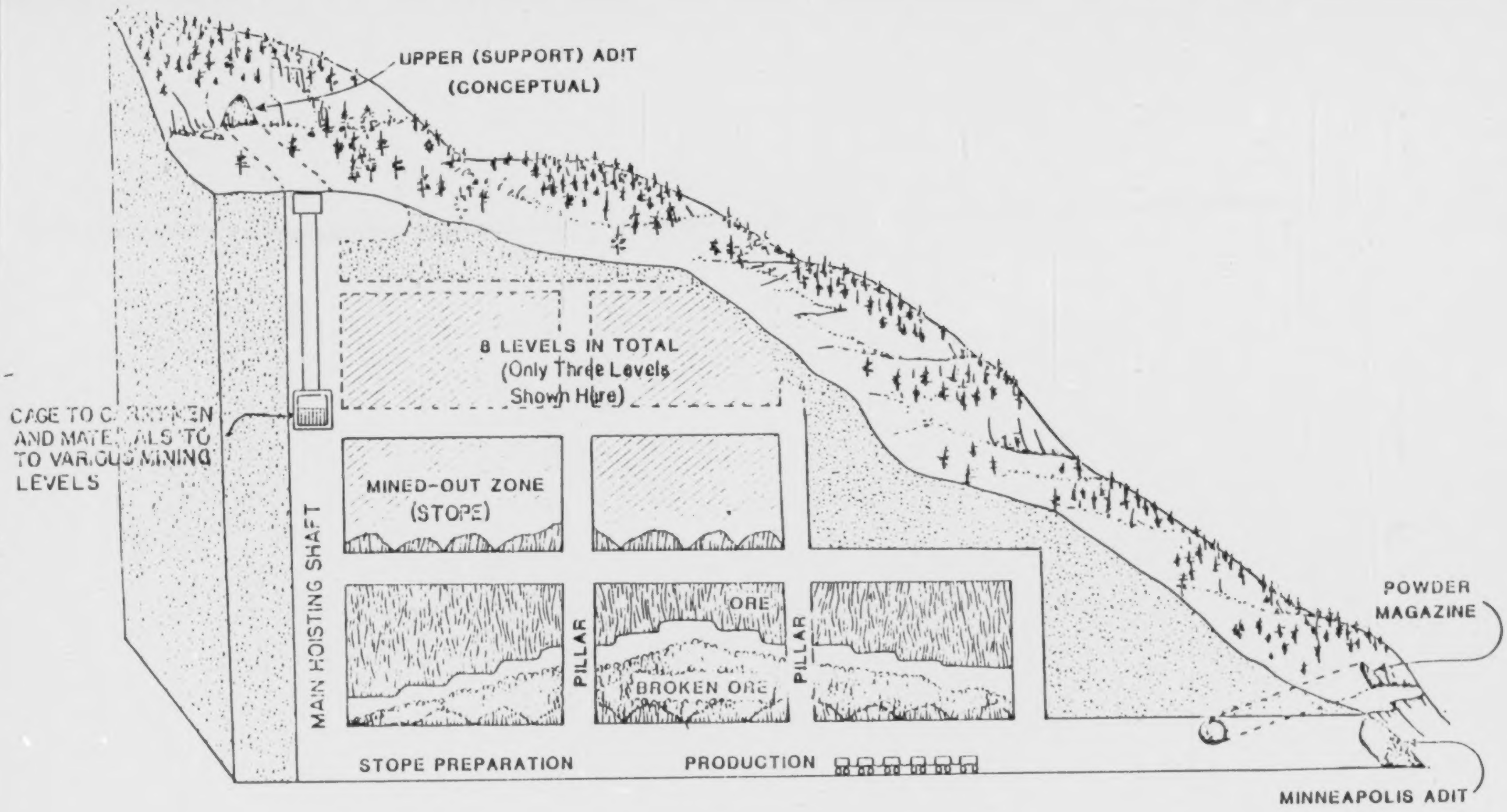

UPPER (SUPPORT) ADIT
(CONCEPTUAL)
8 LEVELS IN TOTAL
(Only Three Levels
Shown Here)
CAGE TO CARRY MEN
AND MATERIALS TO
TO VARIOUS MINING
LEVELS
MINED-OUT ZONE
(STOPE)
MAIN HOISTING SHAFT
PILLAR
ORE
BROKEN ORE
PILLAR
POWDER
MAGAZINE
STOPE PREPARATION
PRODUCTION
MINNEAPOLIS ADIT

FIGURE 5-18-Simplified View of Mining Method

for 350 days per year. A labor force to staff four full shift (weekends included) would be needed. The day shift (Monday through Friday) would normally include 75 employees; the night shifts would include 32 employees. Weekend shifts would also use 32 employees. The number of employees would total about 160 people per day during the week and 125 people per day on weekends.

The company does not contemplate building employee housing. The labor force employed would probably reside in established communities. It is anticipated that the town of Absarokee would accommodate the majority of the people . . . .

The county road (FAS 419 and 420) between the mine and mill would be improved by Anaconda to accommodate the required truck traffic and to provide for public safety. The road would be repaved and bridges would be upgraded as necessary. Widening of the road (other than to accommodate paving) is not anticipated . . . .

The mill is proposed to be located (6 miles from the mine and) . . . the tailing impoundment would be directly to the east and adjacent to the mill . . . .

The mill would operate 24 hours per day, 350 days per year. The staff requirments would be for four full shifts to accommodate weekdays and weekends. The day shift (Monday through Friday) would require ten employees; the night and weekend shifts would be staffed by seven employees per shift . . . .

The finely crushed ore fed into the mill and concentrator would average about 1000 tons per day. Production of concentrate from the mill concentrator would be 15 tons per day. This figure represents a raw milled product containing approximately 0.5 troy ounce of precious metals per ton of ore (500 ounces per day) as well as the associated carrier substances. Roughly 98% of the original ore rock that is processed would end up in the tailing pond and be permanently stored there.

Upon arrival at the mill the ore would undergo secondary crushing prior to conventional grinding. Following grinding, a standard flotation process would be used to extract sulfide minerals containing the precious metals from the bulk of milled material . . . .

A total of 400 gpm of water would be used for the milling process, with 300 gpm being recycled water from the tailing impoundment. The remaining 100 gpm would be lost through seepage and evaporation and would have to be replaced from an alternate source (make-up water) . . . .

Make-up water requirements would be obtained from a combination of a well and a surface water diversion. The surface diversion would be from the West Fork of the Stillwater River via an existing irrigation ditch.

Reagents that would be used for the milling process and their approximate rate of use are copper sulfate (16.7 pounds per day), potassium amyl xanthate (91.1 pounds per day), carboxy methyl cellulose (555.5 pounds per day) and polyglycol ether (as needed) . . . .

The tailing impoundment is designed to contain 7 million tons of material (20 years production at 350,000 tons per year and would provide permanent storage of tailing sent to the tailing pond each day. The surface area of the impoundment would be approximately 113 acres . . . .

Tailing slurry from the mill would be piped to the north end of the tailing impoundment. The slurry would leave the concentrator as approximately 50% solids. The sand-size fraction of the tailing, separated from the rest of the tailing by a cyclone separator, would be used to build the main tailing dam. The finer tailing and slurry water

> removed by the cyclone separator would be piped to the south and discharged into the tailing impoundment . . . .
>
> The distribution line and cylcone separator would be moved back and forth across the tailing dam to raise the elevation of the dam. The top of the tailing dam would remain centered over the starter dam as it is raised. The top of the main dam would stand a maximum of about 100 ft above the preexisting landscape . . . .
>
> After the ore is processed, four truckloads per week of concentrate would be trucked to Columbus for shipment by rail to various market locations . . . .

Proposed reclamation of the mine Complex would include sealing and abandoning all adits, removing facilities, regrading cut and fill slopes, and excavating percolation ponds to handle adit discharges. Fifty-three percent of the waste rock stored at the mine site would be transported to the mill site for use in reclaiming the tailings pond. The remaining waste rock would be regraded, topsoiled and revegetated.

The tailings pond would be reclaimed by covering the surface with about 2 feet of waste rock and then 1.5 feet of topsoil. The surface of the tailings pond would be reshaped into a gentle dome so that all runoff would leave the pond surface and not be impounded. The company plans monitoring of both surface reclamation success and groundwater conditions at the tailings site during the reclamation phase.

Citizen Concerns about Mining Development

In a survey of Stillwater County's population concerning attitudes toward Anaconda Minerals proposed mine and mill, three primary advantages of the project were mentioned by residents.

1. Job opportunities;

2. A boost in the area's economy; and

3. A broader tax base for the county.

Three major disadvantages were also noted:

1. Population influx;

2. Overburdening of the county's public service system; and

3. Environmental problems. (the Montana Department of State Lands and the U.S. Forest Service, 1982)

The observation was made that:

> The residents surveyed clearly felt that job opportunities, local econmic stability, and to a lesser extent, shopping facilities, medical and mental health facilities, and cultural opportunities would be improved by development of the project. The residents also clearly felt that air, water and scenic quality, housing costs and availability, ranching and farming, and friendliness and concern of neighbors would be adversely affected.

Some members of the agricultural community in Stillwater and Sweetgrass Counties have organized as the Stillwater Protective Association (SPA) and the Boulder Valley Protection Association, both affiliates of the Northern Plains Resoruce Council. Although not formally opposing either proposed mining project, but preferring that mining not take place at all, these organizations have insisted on strong environmental protection measures during exploration and proposed mining development in the complex. These organizations generally are considered highly effective, well-organized groups in state conservation affairs.

In its comments ib the Anaconda mining proposal EIS, SPA expressed a general concern about the cumulative impact of mining development in the area:

> A concern of the Stillwater Protective Assosciation (SPA) that is as important as the opening of the proposed mine is the expansion of the mine and the possibility of other mines in the area. The Anaconda mine would be large enough to cause some very real problems, but those problems could pale in comparison with the problems that would arise from a much expanded mining industsry in the area. A haul truck on the road every twelve minutes could turn into a truck every two minutes. A couple of hundred vehicle trips per day to the mine area could turn into a thousand trips per day. 650 new people in a rural area could turn into 1500 new people. Elimination of a bighorn sheep herd could be followed by evacuation of the elk and many deer from the area. A beautiful and quiet rural valley could turn into an industrial valley.
>
> The Anaconda mine will set the precedent for future mining activity in the valley. The location of the mill and tailings pond, the method of mining, the way workers are transported to the mine, and the measures that are taken to protect water and wildlife will undoubtedly dictate the way an expanded or new mining operation would operate. This is evident from, among other things, the precedents set and alternatives chosen for this mine by the exploration permit and exploration activity being conducted by the Anaconda Company. It is, therefore, imperative that the Forest Service and the Department of State Lands use the utmost caution in making decisions on this mine. Problems should not simply be dismissed because they are now relatively infrequent, or benign. It is very likely that they could become frequent and severe. (SPA, 1982, p. 1)

SPA has expressed fears that the relatively small-scale Anaconda proposal will be a first step in large-scale mining in the complex. Given the potential buy-out or joint venture by PGM Resources, the chances are good that the Anaconda mine might ultimately be absorbed into a much larger complex-wide platinum-palladium mine.

One of the SPA's major comments which could affect mining costs concerns alternative mining methods. SPA would like to see the proposed tailings impoundment eliminated or at least decreased in size:

> Cut and fill stoping offers a solution to many of the biggest problems associated with the mine. Similarly, shrinkage stoping modified to include replacement of tailings back in the mine would also solve many of these problems. Either method would eliminate the need to haul ore down a county road. Either would avoid the disturbance to area residents, the threat to people in vehicles on the road, the threat to wildlife that cross the road, the danger to bald eagles feeding on roadside carcasses, and the interference with agriculture in the area. Either would restrict mining impacts to a smaller area and help to preserve the rural nature of the area. Either would also sidestep the problem of having a limited

> tailings disposal area. Most of the tailings would fit into the mined out zones.
>
> The cut and fill method is not discussed seriously enough, and the possibility of refilling shrinkage stopes with tailings was not even mentioned. Cut and fill is dismissed because it could 'significantly increase Anaconda's operating costs.' What is 'significant' in this case--5%? 10%? How does this compare with the likely increase to the county budget of coping with the expense induced by hauling private ore in private vehicles on a county road?
>
> Many of Anaconda's costs could be avoided if some method of replacing the tailings in the mine is used. The county road might not have to be rebuilt. The Hertzler ranch could be sold. Trucks would not have to be purchased and operated and their drivers would not have to be employed. Thousands of tons of waste rock would not have to be hauled down the road and a troublesome reclamation project could be avoided.
>
> The great (though intangible) costs to local people and wildlife would also be substantially reduced. A much more thorough analysis of the costs and benefits of alternative mining methods is badly needed. (SPA, 1982, p. 5)

Anaconda and the state of Montana have resisted the kind of analyses proposed by SPA.

Other concerns include allegedly incomplete analysis of the tailings impoundment and its effects on air and water quality, the extent of Anaconda's commitments to mitigate environmental impacts, and the extent of analyses concerning socioeconomic impacts.

The state of Montana and the Custer National Forest are in the process of responding to all citizen comments on the EIS. Conversations with the EIS team indicate none of the SPA comments will be taken seriously enough to hold up the EIS process nor to jeopardize ultimate approval of Anaconda's permit application.

MADISON MINE/SOUTHEASTERN MISSOURI

The Anschutz Mining Corporation has proposed reopening the presently inactive Madison lead mine at Fredericktown, Missouri, (Figure 5-19). The mine is located in what is known as the "Old Lead Belt," where lead and zinc

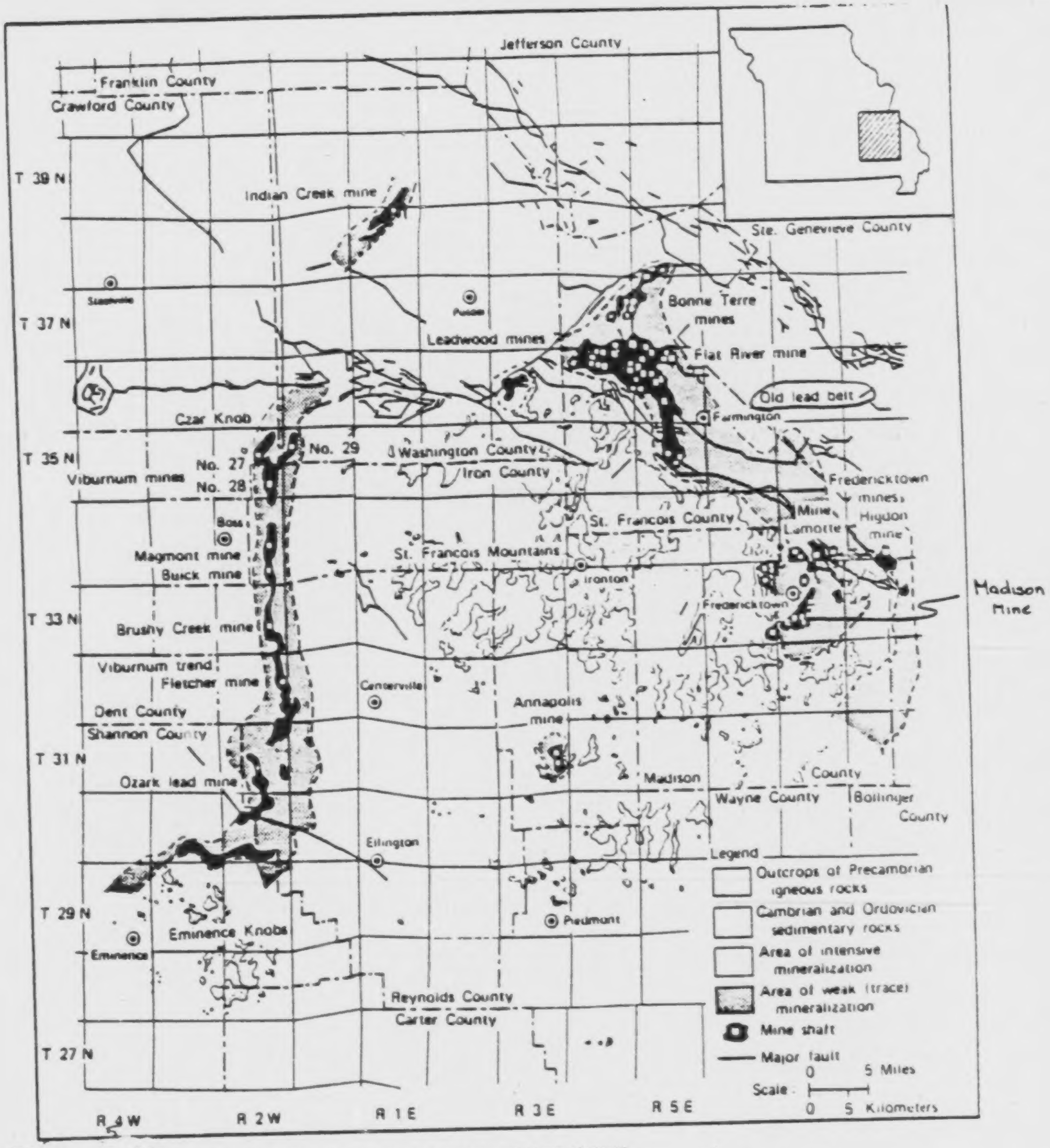

Jefferson County
Franklin County
Crawford County
T 39 N
Indian Creek mine
Ste. Genevieve County
Steelville
T 37 N
Potosi
Bonne Terre mines
Leadwood mines
Flat River mine
Old lead belt
Czar Knob
Farmington
T 35 N
No. 27
No. 28
No. 29
Viburnum mines
Washington County
Iron County
Fredericktown mines
Mine Lamotte
Higdon mine
St. Francois County
Boss
St. Francois Mountains
Magmont mine
Buick mine
Ironton
Madison Mine
Fredericktown
T 33 N
Brushy Creek mine
Viburnum trend
Fletcher mine
Centerville
Annapolis mine
Dent County
Shannon County
T 31 N
Ozark lead mine
Madison County
Wayne County
Bollinger County
Ellington
T 29 N
Piedmont
Eminence Knobs
Eminence
Reynolds County
Carter County
T 27 N
R 4 W
R 2 W
R 1 E
R 3 E
R 5 E
Legend
Outcrops of Precambrian igneous rocks
Cambrian and Ordovician sedimentary rocks
Area of intensive mineralization
Area of weak (trace) mineralization
Mine shaft
Major fault
Scale
0 5 Miles
0 5 Kilometers

Figure 5-19 The southeast Missouri lead-zinc-copper mining district

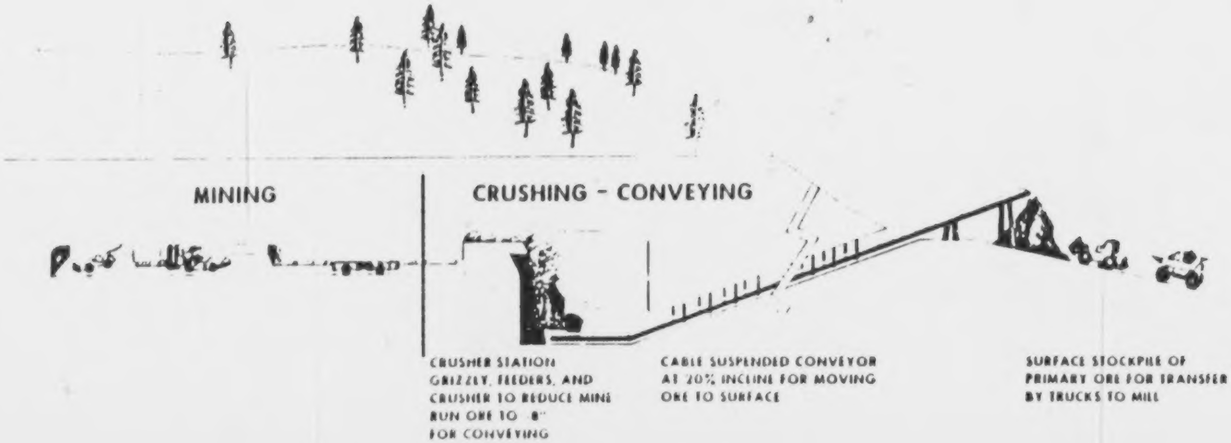
(from Jensen and Bateman, 1979, fig. 19-34) deposits have essentially been mined out. Present lead and zinc mining takes place in a new district about 50 miles west of the "Old Lead Belt" (Jensen and Bateman, 1979). The mine would have an estimated production of two million pounds of cobalt per year; however, a price support of $20.00 to $25.00 per pound is necessary for the mine to become operational (Melbye, 1982). The company had proposed a price support of $28.50 per pound in 1981, but further studies have led to a lowering of the requested price. As part of the project, Anschutz also proposes to construct a refinery which would have the potential of processing ores from lead belt producers. At present, the depressed price of cobalt and the lack of government action on the proposed price supports has led to the "hold" status for the mine. Three employees maintain the facilities and continue dewatering operations. No other activities are underway. The proposal faces no major regulatory or environmental hurdles which would adversely affect the project, including siting of a smelter.

The Madison Mine was first opened in 1847 and operated for two years as a copper mine. The mine also operated from 1860 to 1863, 1901, to 1910, and 1913 to 1920. In the third time period, the mine was operated as the Missouri Cobalt Company. From 1944 to 1961, the mine was operated by National Lead and produced lead, copper, cobalt, and nickel. In 1979, Anschutz acquired an option on the property and began exploration, evaluation, and metallurgical research Melbye, 1981).

The proposed mining plan calls for room and pillar mining for about two-thirds of the production (Figure 5-20). Cut-and-fill techniques would be employed on the other portion of the production. Production will be about 2000 tpd (Amaro, 1980). Old mill tailings would also be processed. Tailings

Figure 5-20

SCHEMATIC MINING SECTION


MINING
CRUSHING - CONVEYING
CRUSHER STATION GRIZZLY, FEEDERS, AND CRUSHER TO REDUCE MINE RUN ORE TO 8" FOR CONVEYING
CABLE SUSPENDED CONVEYOR AT 20% INCLINE FOR MOVING ORE TO SURFACE
SURFACE STOCKPILE OF PRIMARY ORE FOR TRANSFER BY TRUCKS TO MILL

would be "mined" using hydraulic monitors, and the resulting slurry would be piped to a mill feed thickener.

To date, Anschutz has dewatered the mine and rehabilitated the No. 1 shaft collar (Figure 5-21). Further geologic exploration hasresulted in identification of additional reserves beyond the already known block.

Limited pilot plant metallurgical testing has also been completed, including pilot flotation testing and pilot roasting tests on pilot plant concentrates.

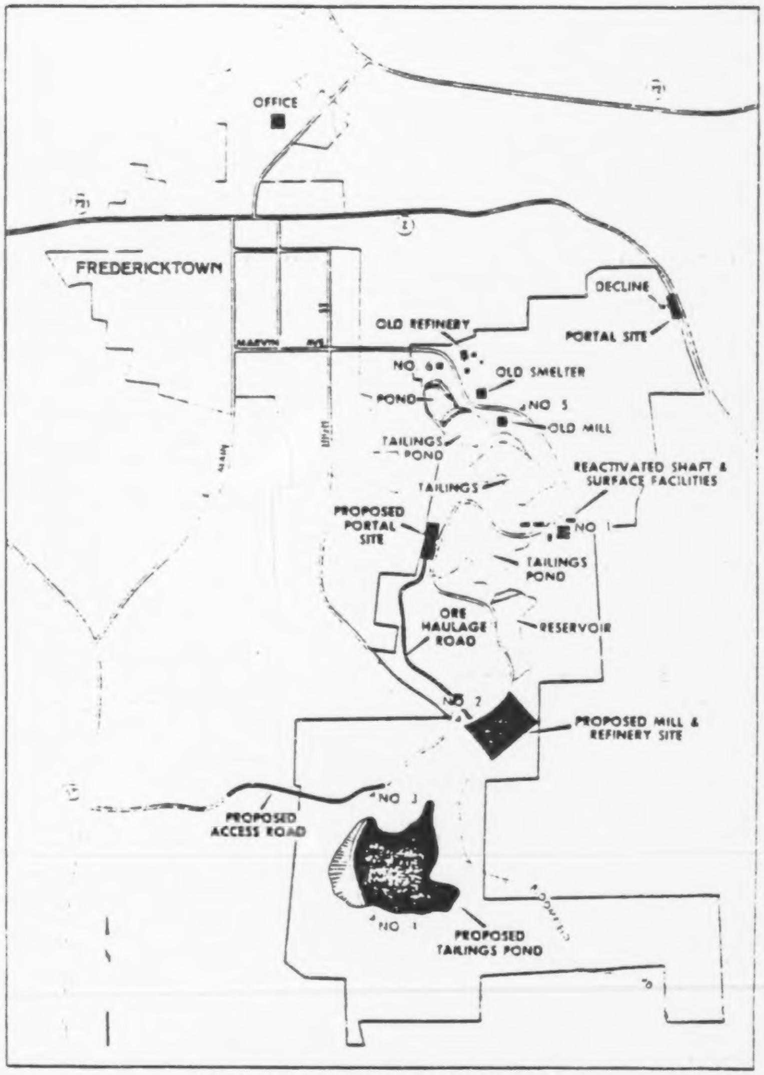
The mine discharges under a NPDES permit administered by the Missouri Department of Natural Resources. An air quality permit would have to be obtained prior to construction of a smelter and the company has alaready collected a year's bsackground data of site meteorology. The company does not believe that obtaining any permits necessary for the project will be difficult in Missouri.

Reserves for the project have been recently been recalculated:

Ore	12.1 MT
Cobalt	22,045 MT
Nickel	31,500 MT
Copper	37,773 MT
Lead	44,409 MT

(Melbye, 1982)

These estimates are for geologic resources, not recoverable reserves. Deposits occur chiefly in the flat-lying Cambrian age Bonneterre Dolomite and lie at shallow depth, within 225 feet of the surface (Figure 5-22). Anschutz has identified a previously unrecognized cobalt ore body, not containing the

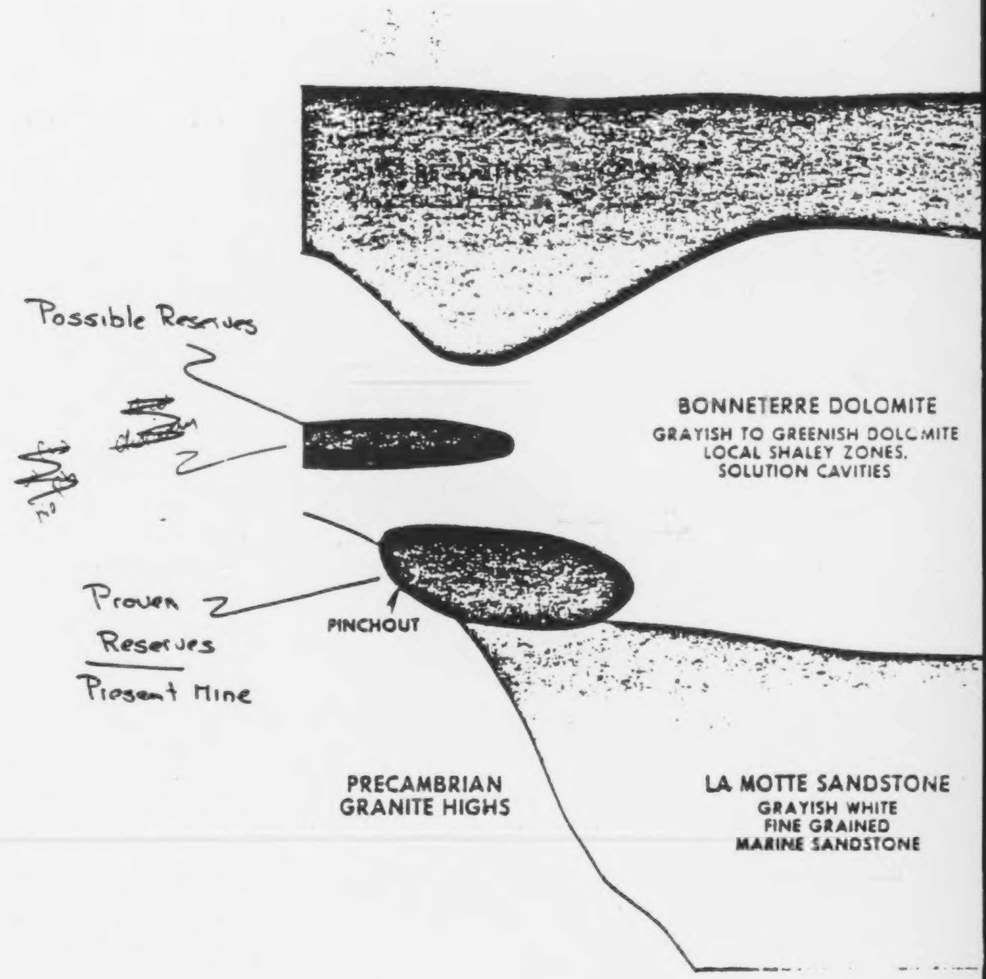

OFFICE
FREDERICKTOWN
MARVIN AVE
MAIN
DECLINE
PORTAL SITE
OLD REFINERY
NO 6
OLD SMELTER
POND
NO 5
OLD MILL
TAILINGS POND
TAILINGS
REACTIVATED SHAFT & SURFACE FACILITIES
PROPOSED PORTAL SITE
NO 1
TAILINGS POND
ORE HAULAGE ROAD
RESERVOIR
NO 2
PROPOSED MILL & REFINERY SITE
NO 3
PROPOSED ACCESS ROAD
NO 4
PROPOSED TAILINGS POND

PROPOSED SURFACE PLAN

Figure 5-21

Figure 5-22

GENERALIZED
GEOLOGIC CROSS-SECTION


Possible Reserves
BONNETERRE DOLOMITE
GRAYISH TO GREENISH DOLOMITE
LOCAL SHALEY ZONES.
SOLUTION CAVITIES
Proven
Reserves
Present Mine
PINCHOUT
PRECAMBRIAN
GRANITE HIGHS
LA MOTTE SANDSTONE
GRAYISH WHITE
FINE GRAINED
MARINE SANDSTONE

customary associated lead and zinc found in other deposits of the region. The cobalt reserves include mill tailings reserve estimates. Recoverable reserve estimates have not been provided beyond the original 9,090 MT estimate (Melbye, 1981). Anschutz believes that possible ore reserves at the project are 25 to 35 million tons.

In its proposals to obtain funding for smelter construction, Anschutz has suggested that up to 1.5 million pounds of cobalt could also be processed from the region's lead mines (Melbye, 1981). Further discussion with the company indicates that such a scenario is optimistic. Half this amount is included in tailing produced by the lead mines and perhaps only 300,000 to 400,000 pounds would actually be recoverable. The other half is included in the smelter feed and at present technologies, increased cobalt recoveries would result in higher iron concentration in lead and zinc recovery (Daniele, 1982).

DULUTH COMPLEX/NORTHEASTERN MINNESOTA

The Duluth Complex is a Precambrian-aged group of layered igneous rocks exposed in northeastern Minnesota. The Complex contains an estimated recoverable reserve of 20 million metric tons of copper, five million metric tons of nickel, 80,000 to 90,000 metric tons of cobalt, and lesser amounts of titanium, platinum, gold, and silver (Minnesota Environmental Quality Board, 1979). Industry interest in the area was greatest from the late 1960's to late 1970's. During that time, AMAX pursued a combined surface and underground operating called MINNAMAX. However, AMAX has now abandoned the project. INCO also considered opening an open pit operation, and those plans have been abandoned.

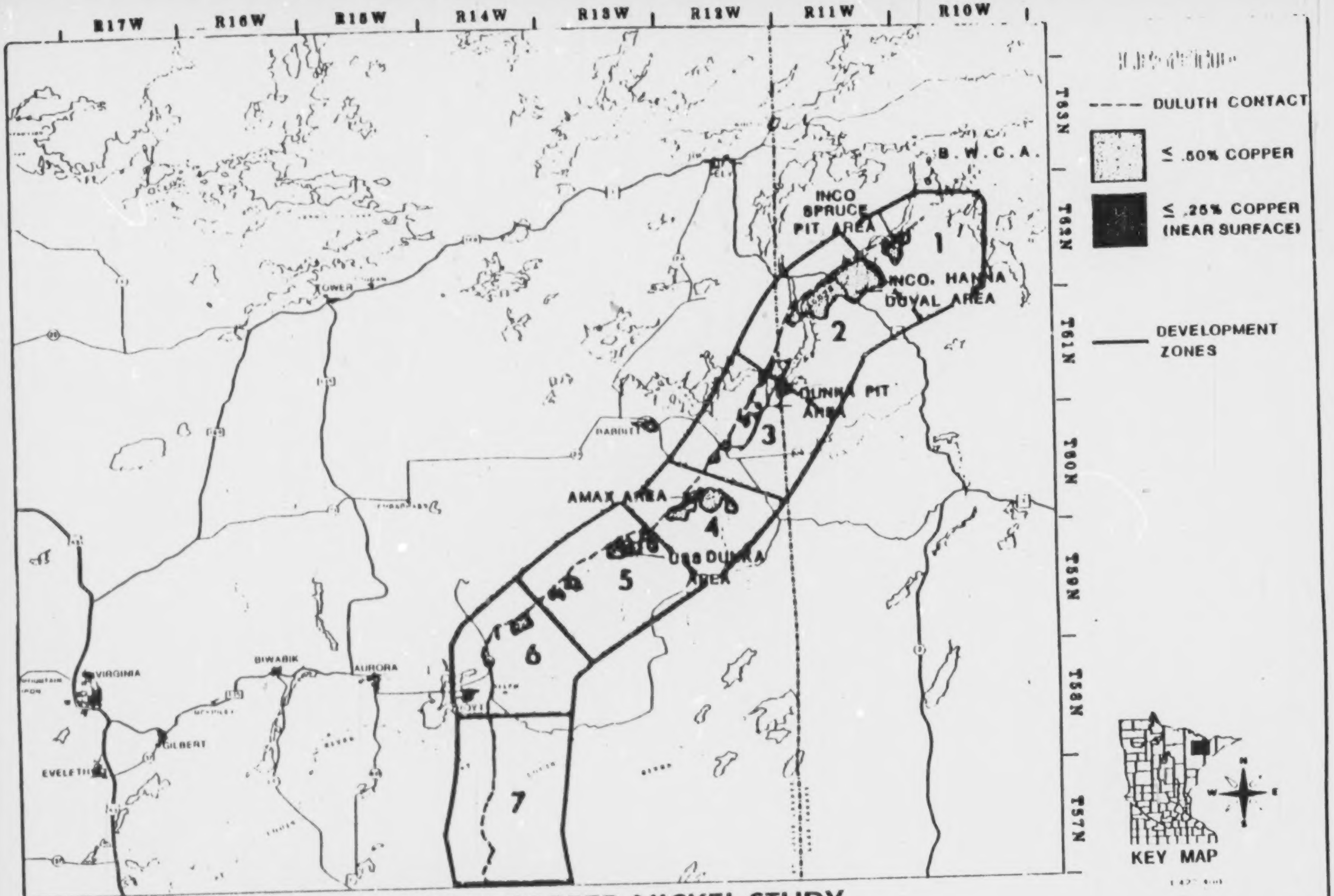
Currently there is no activity in the area; however, the vast resource of low-grade material ensures that northeastern Minnesota will be viewed as a potential source of strategic metals at some time in the future. The proximity of potential mining areas to the Boundary Waters Canoe Area and Voyageurs National Park, as well as the general susceptibility of the northern ecosystem to damage from sulfur emissions from a new smelter needed o process Duluth Complex ores, will also ensure that environmental considerations will play an important role in mine planning in the area.

Environmental Setting

The major copper-nickel deposits of interest in northeastern Minnesota occur in the basal part of the Duluth Complex, in a band 3miles wide and 50 miles long (Figure 5-23). The major land type of the Duluth Complex region is forest, almost all of which is made up of either aspen-birch, spruce-fir, or jackpine stands. Of a 2100 square mile area surrounding the Duluth Complex (referred to elsewhere in this discussion as the "study area"), over three-quarters is in forest. Other land uses of the area include water (8%), taconite mining (3%), bogs and swamps (4%), cultivated and agricultural land (2%), and residential and commercial land (2%). The area as a whole is relatively undeveloped. By the year 2000,, continued taconite mining is expected to convert an additional 1.5% of the area from forest to mining use.

The population of the 2100 square mile area was 51,200 in 1977. Of this number, about 30% were directly employed in the iron industry. Projections made by the state of Minnesota indicate that in the absence of copper-nickel mining in the area, less than 20% of the workforce in 1995 will be involved in the iron industry.

Taconite deposits are found in a southwest/northeast trending area, just


R17W
R16W
R15W
R14W
R13W
R12W
R11W
R10W
T63N
T62N
T61N
T60N
T59N
T58N
T57N
DULUTH CONTACT
≤ .50% COPPER
≤ .25% COPPER (NEAR SURFACE)
DEVELOPMENT ZONES
B. W. C. A.
INCO SPRUCE PIT AREA
INCO. HANNA DUVAL AREA
DUNKA PIT AREA
AMAX AREA
USS DUNKA AREA
1
2
3
4
5
6
7
TOWER
BABBITT
VIRGINIA
BIWABIK
AURORA
GILBERT
EVELETH
KEY MAP

MEQB REGIONAL COPPER-NICKEL STUDY

FIGURE 5-23 MINERALIZATION LOCATION MAP

west of the Duluth Complex. Mining takes place in the Precambrian Biwabik Iron formation, which is stratigraphically below the Duluth Complex. Most communities in the region are located near taconite mining areas and are economically tied to that industry. The northwestern part of the study area, however, is forested and unsettled, except around some lakes. The northeastern part of the study area, and the northern part of the Duluth Complex, is a wilderness and wilderness recreation area.

Present air quality in the area is generally quite good. In undisturbed areas, sulfur dioxide levels are below detection limits and particulate concentrations are low. An exception to this pattern is particulate concentrations around mining areas and population centers, where nonattainment areas have been designated. For purposes of administration of the Clean Air Act, the entire area is designated as a Class II PSD area, with the significant exception of the Boundary Waters Canoe Area and Voyageurs National Park, which are designated Class I areas and subject to more stringent regulations.

In the absence of copper-nickel development, ambient standards for sulfur dioxide are not expected to be exceeded. However, increased SO2 emissions are expected because of the presence of coal-fired power plants and the conversion of the taconite industry to coal, and these changes may limit siting of a new sulfur-emitting source such as a smelter.

Northeastern Minnesota is probably the most susceptible region of Minnesota to the potential environmental impacts of sulfur dioxide and heavy metal air pollution. The pine and aspen vegetation is sensitive to long- or short-term exposure to sulfur dioxide. Soils in the region are relatively thin and poorly buffered against acid input and may be easily damaged by deposition of heavy metals or the influence of acid precipitation.

Surface water in the area is abundant, due to high surface runoff. In the study area evaluated by the state, there are 360 lakes larger than ten acres, in addition to 14 small rivers and streams. Current industrial use of surface water is primarily for electrical power generation and at current levels does not cause significant impacts on the region's water resources.

The quality of the region's water resources is generally very good, except for those areas draining from taconite mining areas. Streams draining undisturbed areas have soft water, with low alkalinity, low total dissolved solids, low nutrients, high color, very low trace metals concentrations, and low fecal coliform counts. A crucial parameter for the region's water resources is their buffering capacity--their ability to regulate pH changes due to acid inputs from precipitation, mine drainage, or other sources. Headwater area lakes and streams are poorly buffered.

Already, the area is experiencing acidic precipitation and there is concern that the ecosystem may already be affected. Fifty percent of precipitaiton sampled in a state study had pH values of 3.6 to 4.4. Studies indicate that the likely sources of acidic precipitation are out-of-state sources as far away as St. Louis, Chicago, and the Ohio valley. Geology

The Duluth Complex is a layered magmatic sediment of Precambrian age. In northeastern Minnesota, it is found in a relatively uncomplicated southeasterly dipping series of formations. The Complex is underlain to the west by the older Virginia formation and the Biwabik Iron formation. Copper-nickel mineralization occurs in the basal part of the Complex and is found along portions of the Complex'snorthern and western contact. Extensive glacial cover precludes surface geological studies in much of the area, increasing the importance of collecting drilling data.

Mineral resource estimates have been prepared by the Minnesota Department of Natural Resources using core assays from 324 drill holes covering a 42.2 square mile area (Minnesota Environmental Quality Board, 1979). Tonnage estimates were made using one of the two following criteria:

1. Resources having a minimum vertical thickness of 50 feet assaying at least 0.5% copper.
2. Resources having a minimum vertical thickness of 100 feet assaying at least 0.25% copper in the top 100 feet of the core (thus amenable to open pit mining techniques). Four major concentrations of resources have been identified (Figure 5-23, Table 5-4):

1. The Spruce Road pit, held by INCO (635 million metric tons of ore resource);
2. The INCO-Hanna-Duval block (2.09 billion metric tons of ore resource).
3. The MINNAMAX area, held by Kennicott (726 million metric tons of ore resource); and
4. The U.S. Steel Dunka Area (272 million metric tons of resource).

Near surface mineralization is estimated to be over one billion metric tons averaging 0.34% copper. The two major near-surface resource areas are the Spruce pit and MINNAMAX areas. Over two-thirds of the resources are deeper than 1000 ft, necessitating underground mining. At MINNAMAX, ore averaged 0.54% Co and 0.13% Ni. Stripping ratio of the INCO property would have been 1.2 and at MINNAMAX 3.

Overall, the ratio of copper to nickel in the Duluth Complex is 3.33:1,

CROSS SECTION

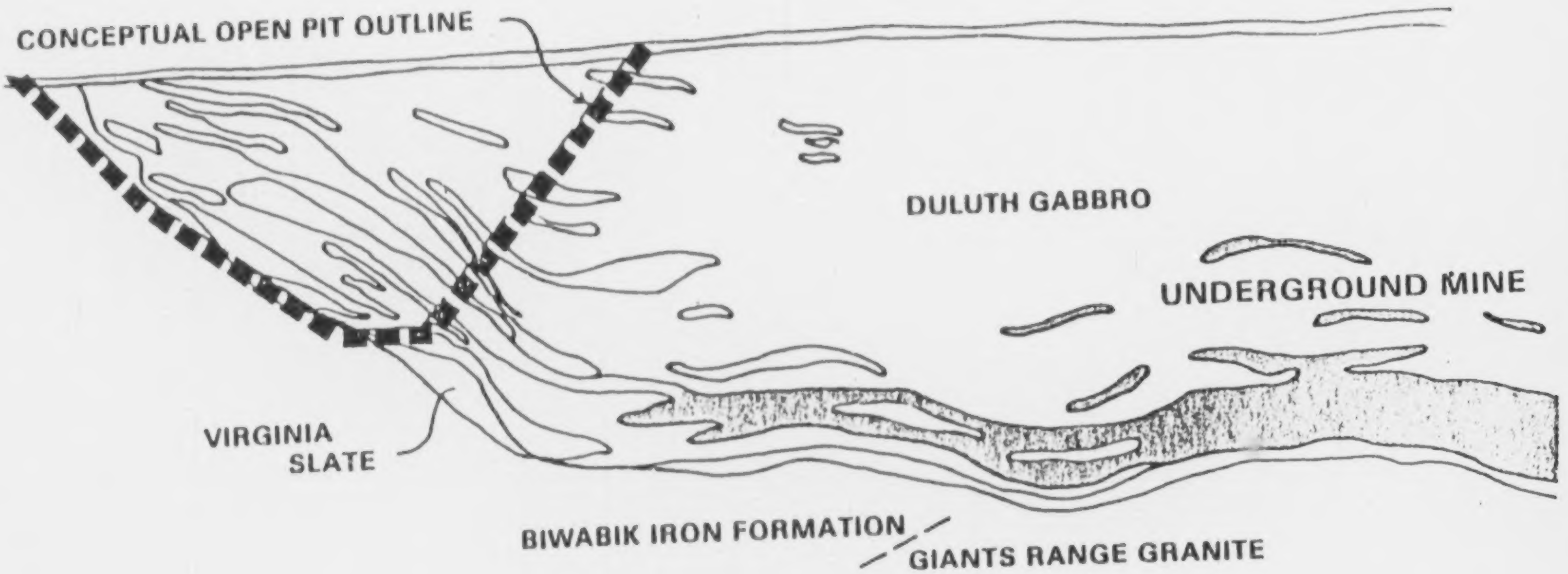

CONCEPTUAL OPEN PIT OUTLINE
DULUTH GABBRO
UNDERGROUND MINE
VIRGINIA SLATE
BIWABIK IRON FORMATION
GIANTS RANGE GRANITE

Figure 5-24

Table 5-4 Copper ore tonnage, grade[a], and metal content estimates, by resource zone.

RESOURCE ZONE	OPEN PIT RESOURCE, million mt 0.25-0.50% Cu Near Surface	OPEN PIT RESOURCE, million mt ≥0.50% Cu Above 1,000 ft	OPEN PIT RESOURCE, million mt Total Above 1,000 ft	UNDERGROUND RESOURCE (million mt) ≥ 0.50% Cu Below 1,000 ft	TOTAL RESOURCE (million mt)	CONTAINED[c] COPPER METAL (million mt)	PERCENT OF TOTAL COPPER METAL
1	110	370	480	370	850	5	18.0
2	---	340	340	1,600	2,000	13	46.8
2&3[b]	15	110	130	-----	130	0.8	2.9
3	250	19	260	76	340	1	3.6
4	180	49	230	550	780	5	18.0
5	38	73	110	230	340	2	7.2
6	60	52	110	49	160	0.9	3.2
7	---	11	11	-----	11	0.07	0.3
TOTAL	650	1,000	1,700	2,900	4,600	28	100

[a] It is assumed that resources of 0.25-0.50% Cu average 0.34% Cu, and ≥0.50% Cu average 0.66% Cu. The numbers have been rounded.

[b] Zones 2 and 3 underwater resources (Birch Lake and the Kawishiwi River) and between 600 ft and 1,000 ft of the surface, assumed to be accessible only by underground methods.

[c] In-place, not recoverable.

but local variations are known. Although copper and nickel are the major metals of interst in the area, several others are also important. Titanium, contained in the mineral ilmenite, could be considered for recovery independent of copper-nickel mining; however, no resource estimates are available. Cobalt occurs in conjunction with copper-nickel deposits and could be recovered as a byproduct. Gold, silver, and platinum could also be recovered.

Based on likely overall recovery percentages for mining and processing, the amount of recoverable copper in the complex is estimated to be about 20 million metric tons, while also yielding 5 million metric tons of nickel. These estimates are based on recoveries .paof 86% for open pit mining and related processing and 68% for underground mining and related processing. Both estimates include smelting and refining.

Mining Industry Interest

Although copper and nickel were found in northeastern Minnesota in the late 1800's, significant interest did not unfold until the late 1940's. In 1978, during construction of a U.S. Forest Service logging road, copper-nickel mineralization was observed. The road was located near Ely. Industry and government began detailed geologic work in the area in the early 1950's, and in the early 1960's exploration activities increased. In the late 1960's, the state of Minnesota began to lease its mineral holdings in the area for exploration purposes.

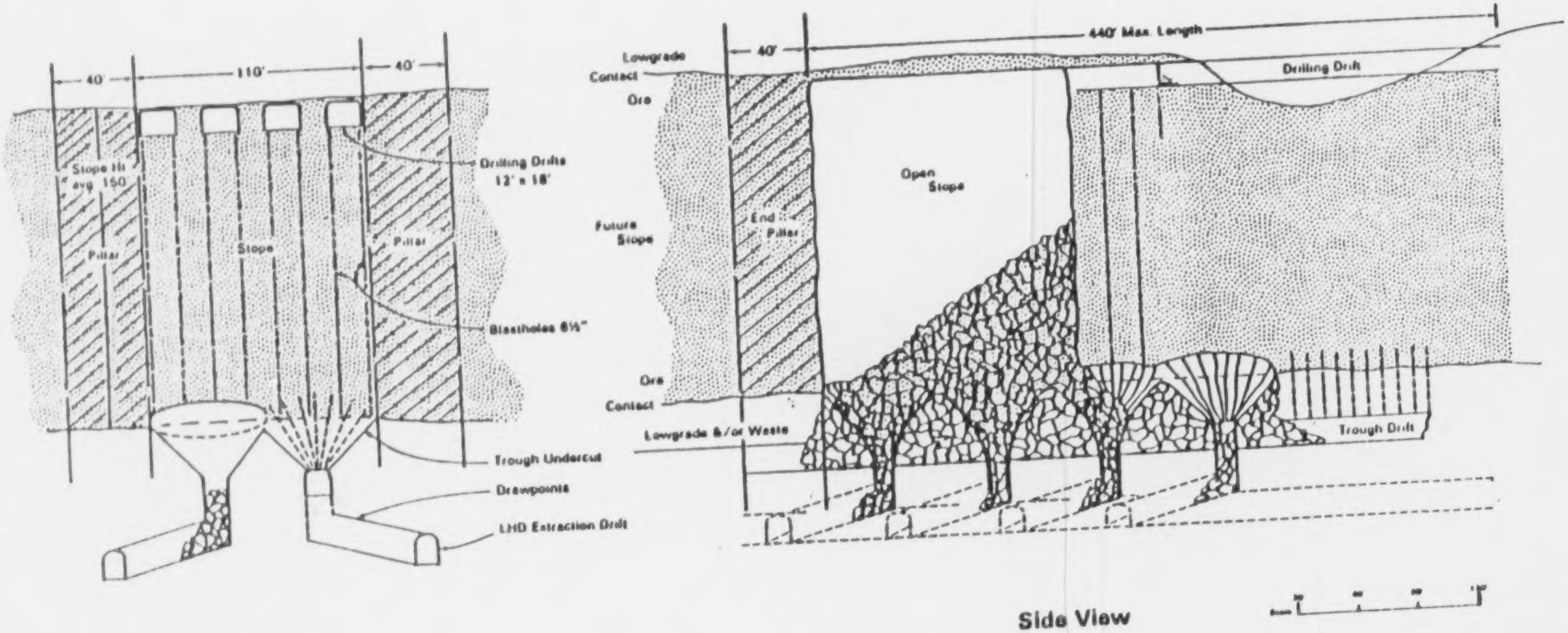
In late 1973, AMAX acquired an option from Kennicott to undertake mine evaluation work on what was named the MINNAMAX project. At the same time, INCO indicated interest in opening a surface mine in the northern part of the complex--the Spruce pit (40,000 tpd ore and 52,000 tpd waste rock). In response to these prospective developments, the state of Minnesota undertook a comprehensive evaluation of the cumulative effects of copper-nickel development. This report cost $4.3 million and was published in 1979 (Minnesota Environmental Quality Board, 1979). In early 1976, AMAX began construction of a test shaft on its prospect, and in June 1977, the shaft reached its desired depth of 1728 ft. Bulk samples were thereafter obtained from the mineralized zone. In 1978, a surface bulk sample was also obtained.

AMAX proposed a combined surface/underground operation (Figure 5-24) employing blasthole open stoping techniques for underground purposes (Figure 5-25). Production would have been 60,000 tons per day, and the life of the operation would have been 40 years (AMAX, 1978). AMAX has now abandoned the project, due to the present unfavorable economic conditions in the mining industry. AMAX could not negotiate an acceptable continuation to its option agreement with Kennicott, and the property has reverted to Kennicott control. Kennicott has no present plans to develop the property (Richards, 1982).

The projected lead times to develop the MINNAMAX project are instructive in projecting the timing of any mining development in the area. Had AMAX continued its development pace after 1978, the company projected that a go/no-go decision on the mine would not have been made until 1985. Obviously, a substantial amount of bulk sampling and geologic evaluation were necessary. Once a decision to mine had been made, another four years were projected before production could begin.

In projecting regional impacts in its study, the state of Minnesota developed likely, or potential, mining models for development throughout the complex. As already noted, open pit and underground techniques are both

BLASTHOLE OPEN STOPING


40'
110'
40'
Stope Ht avg 150'
Pillar
Stope
Pillar
Drilling Drifts 12' x 18'
Blastholes 6½"
Trough Undercut
Drawpoints
LHD Extraction Drift
Lowgrade
Contact
Ore
Future Stope
Ore
Contact
Lowgrade &/or Waste
40'
End Pillar
440' Max. Length
Drilling Drift
Open Stope
Trough Drift

End View

Side View

Figure 5-25

viable for the complex. Based on existing technology and knowledge about the geology of the complex, likely final open pit depths might reach 1000 to 1500 feet. Major ore zones of interest to underground extend from 1000 feet to 3000 feet in depth.

Given the low grade nature of mineralization in the complex, only large-scale operations would allow sufficient economies of scale to permit development. Such large-sale operations, and the unique ratio of copper to nickel in the deposit (3:1), would necessitate construction of a new smelter to process these ores. The state of Minnesota has assumed a flash smelter with a sulfuric acid plant to clean sulfur dioxide from smelter gases.

Environmental Impacts of Development

Several issues have been identified by the state of Minnesota which might affect the character of mine and smelter development in this region. These environmental considerations are an important part of understanding the character of future development in northeastern Minnesota.

One of the most important issues is related to smelter siting. The state has assumed that a smelter would have to be constructed to produce 100,000 metric tons of copper plus nickel per year. Such an operation would generate 200,000 to 400,000 metric tons of SO_2 each year, which would have to be controlled in some way. A series of hypothetical models were created with which to evaluate air quality impacts. The most important conclusions of this modeling analysis are that:

1. Even for the smelter model with the least SO_2 control, the Class II PSD increment is not predicted to be exceeded. Ambient air standards are even higher and thus would not be exceeded. These results, however, do

not also factor in other new SO_2 sources which might use part of the PSD increment.

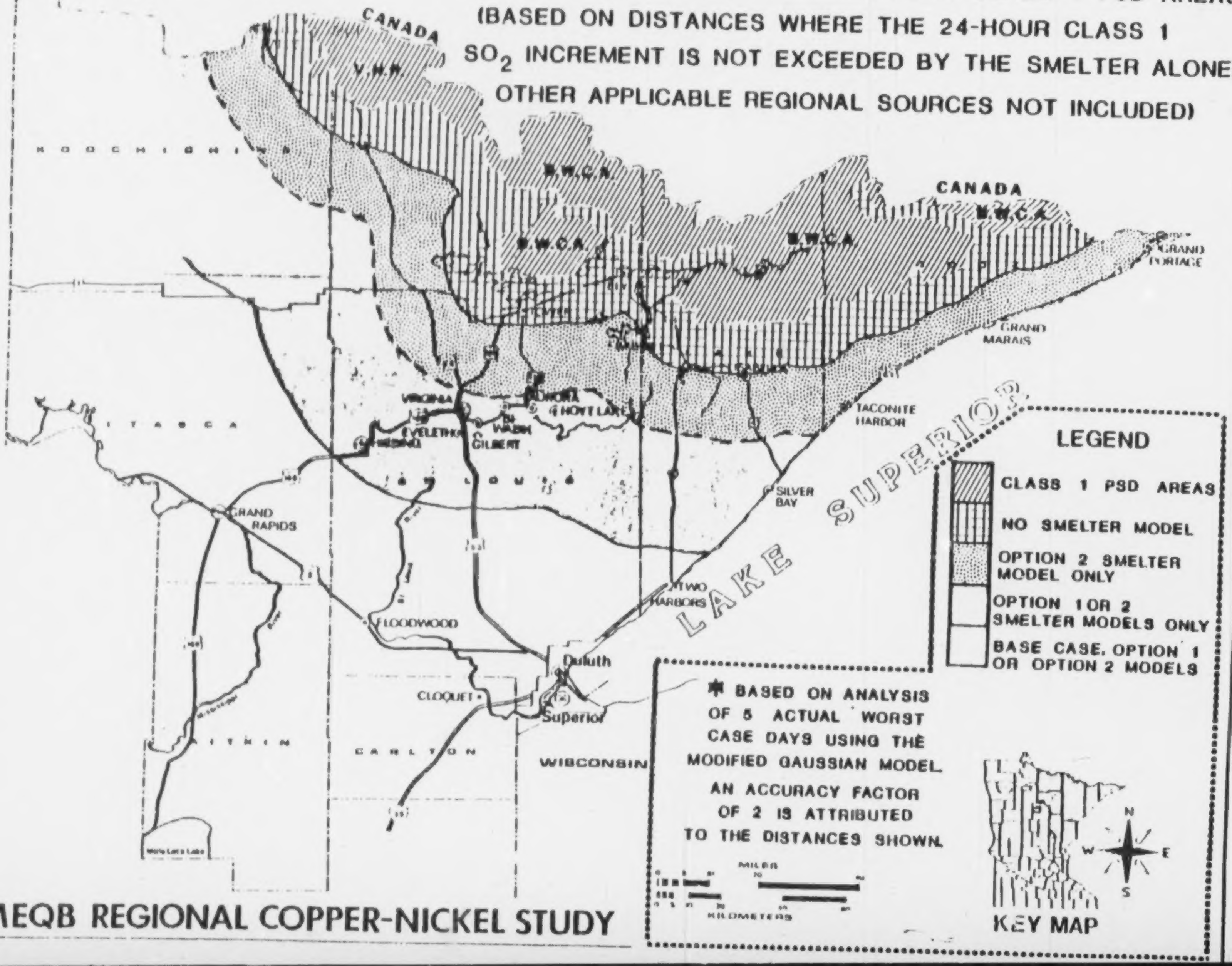
2. In every smelter model, the Class I PSD increment is predicted to be exceeded for a considerable distance from the smelter.

The state then developed a map showing the distance smelters of varying emission control could be sited from the Class I airsheds of the region. The entire Duluth Complex area would be too close to the Class I airsheds to allow smelter construction with only limited control (acid plant control for storing SO_2 gas to 650 ppm SO_2) (Figure 5-26).

Potential particulate emissions from mining operations, particularly open pit mines, also will limit mine siting. In order to not exceed the 24-hour Class I PSD increment for particulates, an open pit mine could probably not be sited within six to nine miles of a Class I airshed. The INCO Spruce pit would be within such a zone of Boundary Waters Canoe Area (BWCA).

The proximity of the Boundary Waters Canoe Area to the Duluth Complex is certainly the most important factor affecting the pattern of mining development in the area. State and federal laws prohibit mining within the BWCA unless Congress declares a national emergency. Although sulfide mineralization is known in BWCA, no estimates are available of the grade or tonnage of deposits. In part, this is due to limitations on exploration in the area. The boundaries of the BWCA were recently extended to include a portion of the northern end of the Duluth Complex, as well as establishing a Mining Protection Area, consisting of several road corridors adjacent to the BWCA where mining is also prohibited. This mining protection area does not include any of the Duluth Complex, but it does include a portion of the Greenstone formations, near Ely. The Class I airshed status of the BWCA poses

FIGURE 5-26 SMELTER SITING ZONES* SURROUNDING CLASS 1 PSD AREAS (BASED ON DISTANCES WHERE THE 24-HOUR CLASS 1 SO_2 INCREMENT IS NOT EXCEEDED BY THE SMELTER ALONE, OTHER APPLICABLE REGIONAL SOURCES NOT INCLUDED)


CANADA
V.N.P.
KOOCHICHING
B.W.C.A.
B.W.C.A.
B.W.C.A.
CANADA
B.W.C.A.
GRAND PORTAGE
GRAND MARAIS
ITASCA
VIRGINIA
EVELETH
HIBBING
GILBERT
AURORA
HOYT LAKES
TACONITE HARBOR
ST. LOUIS
GRAND RAPIDS
SILVER BAY
LAKE SUPERIOR
TWO HARBORS
FLOODWOOD
Duluth
CLOQUET
Superior
WISCONSIN
AITKIN
CARLTON
Mille Lacs Lake
LEGEND
▨ CLASS 1 PSD AREAS
▥ NO SMELTER MODEL
░ OPTION 2 SMELTER MODEL ONLY
☐ OPTION 1 OR 2 SMELTER MODELS ONLY
☐ BASE CASE, OPTION 1 OR OPTION 2 MODELS
MILES
KILOMETERS
N
E
S
W
KEY MAP

* BASED ON ANALYSIS OF 5 ACTUAL WORST CASE DAYS USING THE MODIFIED GAUSSIAN MODEL. AN ACCURACY FACTOR OF 2 IS ATTRIBUTED TO THE DISTANCES SHOWN.

MEQB REGIONAL COPPER-NICKEL STUDY

limitations on mining and smelting development. Dust from mining would have its greatest effect on the BWCA if it occurred in the northern part of the complex, where 75% of the resources are known to be located. A smelter should not be sited within six to 12 miles of the BWCA to "guard against significant risks of damage during breakdown conditions" (Minnesota Environmental Quality Board, p. 71).

Discharges of surface water to the BWCA could also harm the ecosystem of the poorly buffered waters. A high impact potential area has been identified at the north end of the complex (Figure 5-27). The INCO Spruce pit falls into a zone of "medium impact potential" because mine drainage would still flow towards the BWCA. The state analysis concluded:

> In summary, federal and state laws prevent copper-nickel development within the BWCA, but significant indirect impacts could occur from copper-nickel (and other development) located outside. Specific criteria for the protection of the BWCA from such impacts do not exist at the state and federal levels, except for some air pollutants. The lakes, streams, and forests of the BWCA are presently in jeopardy because of acid rain (probably caused by air pollution sources throughout the midwest). If the precipitation in the region continues its trend of increasing acidity, then the impacts on the BWCA caused by large-scale copper-nickel development will be small in comparison (both in magnitude and area affected). A 'no impact' requirement for the BWCA, if implemented, probably precludes a smelter in the Study Area (even if best available pollution control systems are used). Copper-nickel development would still be possible because a smelter can be located elsewhere. Still, a no impact requirement would probably preclude any copper-nickel mining in zone 1, and open pit mining in zone 2. Noise from underground operations in zone 2 could cause a a problem, but noise suppression techniques should be able to correct this situation. (Minnesota Environmental Quality Board, 1979, p. 73-74)

The state estimated that a 10% to 15% savings would result from location of a smelter at the same location as a mine and mill Complex. Such a savings could not be realized if air quality considerations prevented siting of a smelter on or adjacent to the Duluth Complex. The state analysis does not discourage smelter siting elsewhere in Minnesota, and observes, "Certainly an

120B

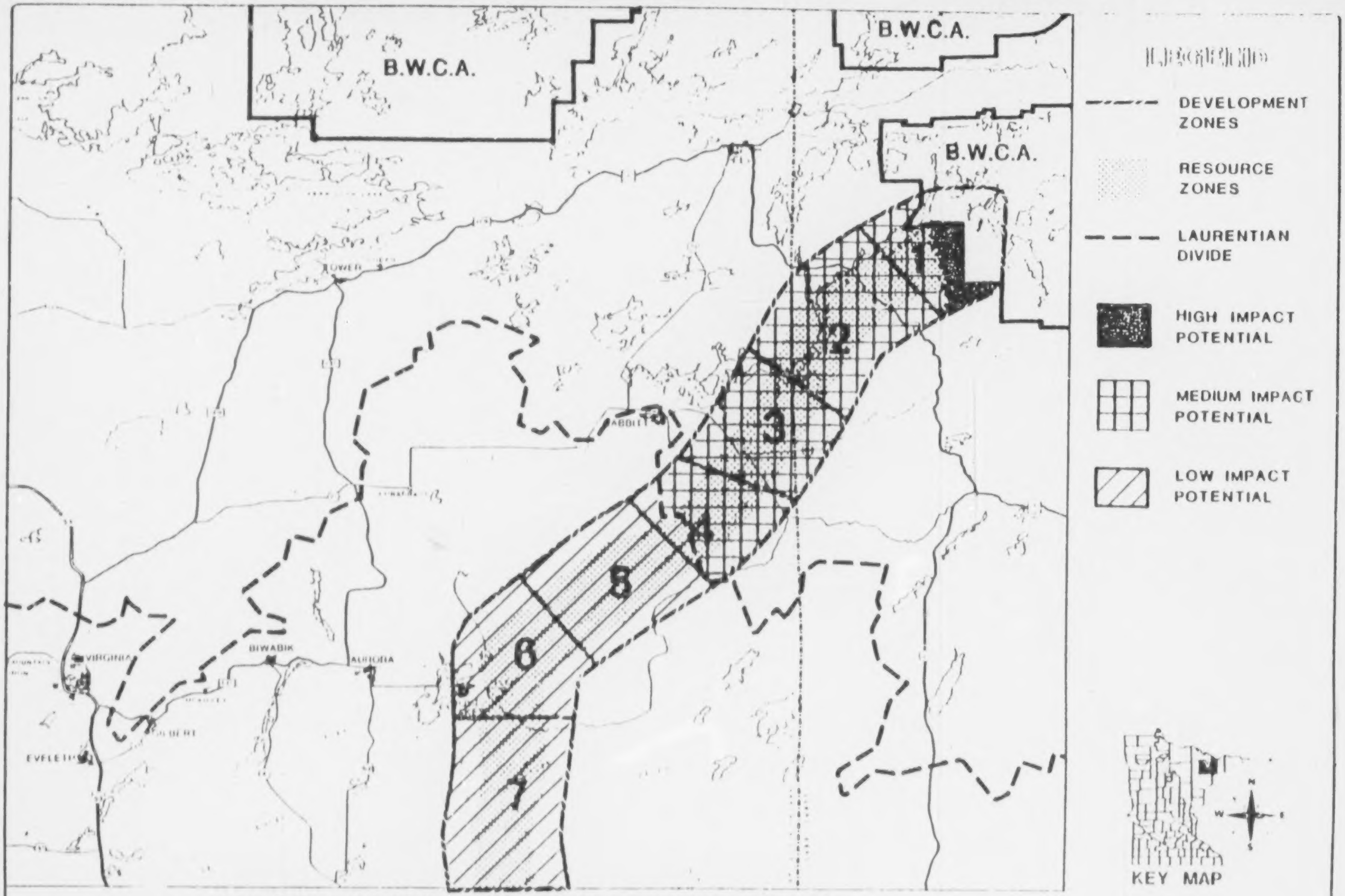

B.W.C.A.
B.W.C.A.
B.W.C.A.
LEGEND
DEVELOPMENT ZONES
RESOURCE ZONES
LAURENTIAN DIVIDE
HIGH IMPACT POTENTIAL
MEDIUM IMPACT POTENTIAL
LOW IMPACT POTENTIAL
1
2
3
4
5
6
7
TOWER
ABBIT
BIWABIK
AURORA
VIRGINIA
GILBERT
EVELETH
N
S
E
W
KEY MAP

FIGURE 5-27 MEQB REGIONAL COPPER-NICKEL STUDY

B.W.C.A. WATER QUALITY IMPACT POTENTIAL OF DEVELOPMENT ZONES OUTSIDE THE B.W.C.A.

industrial location and population center, such as Duluth, would not be as sensitive to the added metals buildup in the soils or acidification effects from a smelter" (Minnesota Environmental Quality Board, 1979, p. 90-91).

Hydrologic considerations may affect mine siting, although the potential for mitigating these impacts is greater than for air quality concerns. The dilemma for mine siting is that minimizing hydrologic impacts would dictate siting in the southern part of the Complex, where waters do not drain to the BWCA. However, 75% of the resources lie north of the divide, where waters drain to the BWCA. The entrepreneurial nature of mining development in the United States does not include government siting of mines, and so there is little way that mining can be encouraged in the southern part of the Complex. It is clear, however, that mining proposals in the north, such as the INCO Spruce pit, would be subjected to a more intense regulatory review, and it is possible that mining might be prohibitied in that area.

In contrast, the MINNAMAX project area, lying 15 to 20 miles south of the BWCA, is a more desirable mine site from an environmental standpoint. The site's location on the watershed divide between north- and south-flowing waters also provides a better potential for control of contaminated waters.

Table 5-5 summarizes estimated impact to resources from these constraints. In Zone 1, 1.5 million MT of nickel and 30,000 MT of cobalt might be prohibited from mining, while 20 percent of Zone 2 resources (those surface mineable) could also be affected. Mining in Zones 3 and 4 would have to utilize careful environmental planning. Thus, interplay between environmental and mineral interests is a factor in mining development in northeastern Minnesota. If global mineral economic conditions continue to delay the viability of mining development in the region, it is possible that

Table 5-5

Environmental Considerations and Mineral Resources - Duluth Complex

CONTAINED METAL

Resource[1] Zone	Copper[2] Content	Nickel[3] Content	Cobalt[4] Content	Air[5] Impacts
1	5	1.5	0.03	high
2	13	3.9	0.07	
2&3b	0.8	.2	0	
4	5	1.5	0.03	
5	2	.6	0.01	
6	0.9	.3	0.01	
7	0.1	0	0	

[1] Figure 5-23

[2] Table 5-4, column 7

[3] Based on constant Cu/Ni ratio 3.33:1

[4] Based on constant Co/Ni ratio .017:1

[5] See text

new technologies, some of which are discussed in Chapter VII, in mining, reclamation, water quality management, and air pollution control may minimize projected impacts of development not proposed for another 10 or 20 years.

REFERENCES

Goodnews Bay

Bundtzen, T.K., Eakins, G.R., and Conwell, C.N., 1982, Review of Alaska's mineral resources: State of Alaska, Department of Commerce and Economic Development, Office of Mineral Development, 30 p.

Carnes, R.D., 1981, Platinum Group Metals, Salmon River, Alaska: U.S. BM Deposit Summary Report.

Kerwin, Pete, 1982, Personal communication: Hanson Mining Co., telephone interview with J. Schmidt, December 6.

Nickel Mountain

Johnson, M., 1982, Personal communication: interviewed by A. Silverman, August 19, 1982.

Ramp, Len, 1978, Investigations of nickel in Oregon: Oregon Department of Geology and Mineral Industries Miscellaneous Paper 20, 68 p.

Ridenour, J., et al., Nickel Mountain: USBM deposit summary report.

Blackbird Project

Becker, K. (Salmon National Forest), 1980, Personal Communcation: interview with J. Duncan, ERT, September 16, 1980.

Bennett, E.H., 1977,Bilbrey, J.H., 1962, Cobalt, a materials survey: U.S. Bureau of Mines Information Circular IC-1803, 140 p.

Buchta, Tom (minerals specialist, Cobalt Ranger District), 1982, Personal communication: telephone interview, November 17, 1982.

Cobalt Ranger District, 1982, Environmental assessment, Sage Project, 12 p.

Erickson, Lill, 1981, Letter to Jim Lancaster, Cobalt District Ranger: November 12, 3 p.

Fiorini, R.J., 1981, Letter to Sen. Harrison Schmitt: in Defense Production Act and the Domestic Production of Cobalt, Hearing before U.S. Senate Committee on Banking, Housing, and Urban Affairs, p. 143.

Peters, Stephen G., 1981, Final geology and mineral resources technical report for the Blackbird project: Noranda Mining report to Salmon National Forest.

Platts, W.S., Martin, S.B., and Primbs, E.R.J., 1979, Water quality in an Idaho stream degraded by acid mine waters: USDA General Technical Report INT-67.

Salmon National Forest, 1982, Final Environmental Impact Statement, Blackbird cobalt-copper project: Salmon, ID.

Scheuering, Joe (General Superintendent, Blackbird Mine), 1982, Personal communication: interview, September 24, 1982. U.S. Environmental Protection Agency, 1980.

Gasquet Mountain

Berthelsen, J., 1982, Cobalt on Gasquet Mountain: Sierra, November/December, p. 55-58.

Kaiser Engineers, 1982, Mine plan, Gasquet Mountain: report to California Nickel, March 17.

Sassenrath, C. (official of California Air Resources Board), 1982, Personal communication: telephone interview with J. Schmidt.

U.S. Forest Service, 1983, Draft of environmental impact statement, Gasquet Mountain project: Six Rivers National Forest and Del Wade County.

Stillwater Complex/Beartooth Face

Hottl, John B., 1954, How one chrome producer aims at self-sufficiency: Engineering and Mining Journal, v. 155, June.

Montana Department of State Lands and Custer National Forest, 1982, Draft environmental impact statement, Anaconda Stillwater project.

Read, R.F., 1977, Summary report-mineral development potential of the Stillwater Complex, Montana: Spokane, Western Field Operations Center, U.S. Bureau of Mines.

Stillwater Protective Association, 1982, Comments on the draft environmental impact statement for the Anaconda Minerals Company Stillwater Project: 14 p.

Wimmler, N.L., 1948. Investigation of chromite deposis of the Stillwater Complex; U.S. Bureau of Mines report on investigations, 41 p.

_______ 1982, "PGM Resources puts mining projection hold", Big Timer Pioneer, June 9.

Northern Plains Resource Council, 1982, "Anaconda Delays Mine Decisions": The Plains Truth, July.

U.S. Forest Service, 1978, Environmental statement, Beartooth Face planning unit: report USDA-FS-Adm-Fes-01-08-77-11, 191 p.

Madison Mine

AMAX, 1978, MINNAMAX copper-nickel evaluation project, federal/state/AMAX meeting briefing packet: 7 p.

Daniele, Dick (Vice-President, Anschutz Mining), 1982, Personal communication, telephone interview with J. Schmidt, December 23.

Jensen, Mead L., and Bateman, Alan M., 1979, Economic mineral deposits: New York, John Wiley and Sons, 593 p.

Melbye, Charles (President, Anschutz Mining), 1982, Personal communication, telephone interview with J. Schmidt, December 23.

Melbye, Charles, 1981, Testimony in Defense Production Act and the domestic production of cobalt, hearing before the Senate Committee on Banking, Housing, and Urban Affairs: Senate Report 97-38, pp. 144-193.

Minnesota Environmental Quality Board, 1979, The Minnesota regional copper-nickel study, 1976-1979, executive summary: St. Paul, 93 p.

Chapter VI

PROCESS TECHNOLOGY FOR DOMESTIC SOURCES OF CHROMIUM, COBALT, MANGANESE, NICKEL, AND PLATINUM

This chapter outlines current process technology options for the commodities of interest in this study. The chapter focus is existing options for processing these deposits and improvements available in the near-term. In some cases, recommendations are offered concerning investment and development of selected processing options. Each section is prefaced with conclusions specific to that commodity. Implications of these recommendations are discussed in Chapters VIII and IX.

CHROMIUM

Introduction

As discussed in Chapter II, two important sources of domestic primary chromium are the Montana Stillwater chromites and the disseminated California and Oregon ultramafic chromites. There is also chromite associated with the Ni-Co laterite ores of northern California and southern Oregon. Details of process metallurgy for chromite and ferrochrome recovery are discussed below. Process metallurgy for chromite recovery from domestic laterite will be examined briefly. The economic viability of recovery from laterites is particularly questionable due to the large amount of magnesium, aluminum, and iron incorporated within the lateritic chromite mineral structure.

The world's primary chromium resource occurs in the form of the mineral chromite ($FeO\ Cr_2O_3$), which theoretically contains 68% chromic oxide (Cr_2O_3). However, chromite usually contains less than 68% Cr_2O_3 because the chromium and iron may be replaced by varying amounts of aluminum and magnesium, respectively. Metallurgical-grade chromite contains 46% Cr_2O_3 and has a Cr:Fe weight ratio of at least two (Smith, 1980). High Cr/Fe ratio is desirable to minimize the volume of ore treated, to minimize the amount of iron that must be reduced in smelting and to maximize the Cr/Fe ratio in the product ferrochromium. Good smelting ores contain about equal portions of Al_2O_3 and MgO, with those having higher Al_2O_3 contents being preferred. Low silica is desirable (e.g., 5% SiO_2), but many ores contain 10% or more silica, principally in the form of serpentine (i.e., magnesium silicate). Silica, magnesium, and alumina increase the volume of ore that must be melted; and if present in the wrong ratios, can necessitate the addition of high flux addition (i.e., slag additives) in order to avoid formation of high-melting-point viscous slag.

Chemical-grade chromite contains 40% to 46% Cr_2O_3 and has a Cr/Fe ratio of 1.5 to 2.0. Refractory-grade chromite contains more than 20% Al_2O_3 and more than 60% Al_2O_3 plus Cr_2O_3 (Morning, 1980).

Chromite ores are generally cleaned by hand sorting, washing and screening, or gravity concentration. Concentrates are prepared from fines or size-reduced lower grade ore using jigs, tables, spirals, or magnetic separators. Although flotation separation is also used, gravity separation is predominant (Morning, 1980).

For steel production, chromite is converted to chromium alloys or chromium additives by reduction in an electric furnace. For chemical use, the chromite is generally roasted with soda ash and lime, followed by water leaching. This leach liquor is converted to either dichromate solution or a crystalline product. For refractory manufacture, sized chromite is mixed with magnesite (i.e., $MgCO_3$), pressed into refractory shapes, and fired.

Conclusions

1. Of the five strategic metals reviewed (Ni, Co, Pt, Mn, Cr), chromium would benefit most from government-sponsored efforts to improve its availability from domestic sources. Little has been done to target recent developments in beneficiation and smelting on these resources.

2. Recent developments in beneficiation include commercial availability of efficient high-intensity magnetic separators, and techniques for flotation of fine particles. Breakthroughs in smelting technology (e.g., transfer-arc plasma, PAMCO-hot-alloy steel making) permit use of lower grade and finely-ground chromite.

3. Because of the dispersed nature of data on chromium resources and processing there is a clear need for synthesis of available information and "pooling" of experts' ideas, similar to the synthesis-prepared by the NMAB (1976) for manganese.

Stillwater Chromite

The three largest chromite deposits in the Stillwater Complex of Montana are the Mouat, Benbow, and Gish. Anaconda, under contract to the U.S. Government, began mining the Benbow and Mouat deposits in 1941, but ceased operation in 1943. From 1954 to 1964, 980,000 tons of chromite (36% to 38% Cr_2O_3; Cr/Fe = 1.5) were stockpiled by American Chrome.

A primary jaw crusher reduced American Chrome ore to minus 3-inch. A cone crusher set to yield a one-half- to three-quarter-inch product provided feed to two rod mills equipped with 3-inch rods. The minus 20-mesh product (particles smaller than 830 microns diameter), produced via closed-circuit screening, fed a series of Wilfley tables. Following retabling of coarse table middlings, tabling of overflows from dewatering, and hydraulic classification, a final concentrate was trucked to the stockpile (Huttl, 1954). The chromite flotation technology developed for Mouat ore by the USBM (Batty, 1947) was not utilized in American Chrome's mill, presumably because American Chrome's gravity methods reportedly gave higher grade (38% Cr_2O_3), higher recovery (80%), and avoided using fluoride in an acidic system. Further work on chrome flotation was published by Sullivan (1960) on processing the feed to the fine table section of the American Chrome mill. Tailings from these tables accounted for half of the mill's chromite losses (i.e., 10% of the chromite). Removal of the minus 9-micron fines from the fine-table feed, followed by conditioning with hydrofluoric acid ($NaF + H_2SO_4$) and a fuel oil-tall oil mixture, permitted flotation of 75% of the chromite at 29% grade. Loss of Cr_2O_3 to the minus 9-micron fines (i.e., the slimes) was 4%. Consumption of NaF, H_2SO_4, and oil were 4, 18, and 35 lb/ton, respectively.

Recovery of high-grade chromite concentrate is made difficult by the impure nature of the chromite mineralization. Pure chromite, FeO Cr2O3, consists of ferrous and chromic oxides chemically united in equal molecular proportions in the chromite molecule (also written as $FeCr_2O_4$). Unfortunateny, such purity is seldom realized. In most chrome ores, the actual mineral is a combination of chromite and spinel, MgO Al_2O_3. The ratios between the four common constituents (i.e., FeO, MgO, Cr_2O_3, and Al_2O_3) vary greatly. Other minor constituents can also be present, such as TiO_2 and Fe_2O_3. Pure chromite contains 68% Cr_2O_3, but the Cr_2O_3 content of the chrome minerals in commercial ores can vary between 45% and 60%.

Interest in flotation concentration of Stillwater chromite is still very much alive. The USBM demonstrated that Stillwater chromite floats well, using a cationic collector (0.4 to 1.0 lb/ton of Armac C) in an acidic slurry (20 to 50 lb H_2SO_4/ton) from which fine particles had been removed (Smith et al., 1981). Chromite losses to the slimes were about 5%, which is comparable to Sullivan's 1960 efforts. Ninety percent of the Cr_2O_3 content of the flotation feed (40% Cr_2O_3 gravity concentrate) was recovered into a product containing 45% Cr_2O_3. Smith's goal was presumably to attain the 45% Cr_2O_3 specification for metallurgical-grade chromite. Fundamentals of the flotation of Stillwater chromite have also been studied (Smith, 1982). Flotation response was affected by whether the ground mineral had been aged in air, which apparently leads to surface oxidation. Acidic flotation using dodecylsulfate ($C_{12}H_{24}SO_4$) as a collector showed little aging effect on chromite recovery at the optimum pH 3 to 4. However, aging below pH 3 decreased recovery and above pH 4 increased recovery about pH 4. Pretreatment of the chromite with HCl resulted in the mineral behaving as a simple oxide, with cationic flotation possible only at pH values more basic than the mineral's zero point of charge. Of the gangue minerals studied, olivine was the one most readily floated and this proved to be the most troublesome. As compared to gravity methods, the flotation methods developed by the USBM appear to fall short for upgrading domestic chromite. Acid consumption is high, and the method requires construction of acid-resistant cells. Furthermore, unlike gravity concentration methods, flotation has to contend with residual chemical reagents in the mill-water bleed. A sample of Mouat chromite taken from the American Chrome mine near Nye, Montana, was examined by the USBM to determine the purity of the chromite mineral. Their assays showed that the mineral contained 44% Cr_2O_3, 18% Al_2O_3, 18% Fe, and 12% MgO (Sullivan, 1964). It is thus not surprising that attaining 46% Cr_2O_3 product via physical beneficiation was never achieved.

California-Oregon Disseminated Chromite

The disseminated chromites of California and Oregon are of a lower grade (typically 4% to 5% Cr_2O_3) than the 20% Cr_2O_3 reserves of the Stillwater Complex. However, there is increasing evidence that the total reserves of these lower-grade deposits may be relatively large. The Pilliken deposit (El Dorado County, California) was mined during World War I, and again from 1937 through 1942 (Wetzel, 1980). During a four-month period in 1941, the Rustless Mining Corp. produced 11,000 tons of chromite concentrate (42% Cr_2O_3) from ore averaging 10% Cr_2O_3 that was collected from localized high-grade pockets. Their gravity flowsheet utilized a jig, a hydraulic classifier, and concentrating tables. By 1960, over 39,000 tons of chromite concentrate had been shipped from the Pilliken deposit.

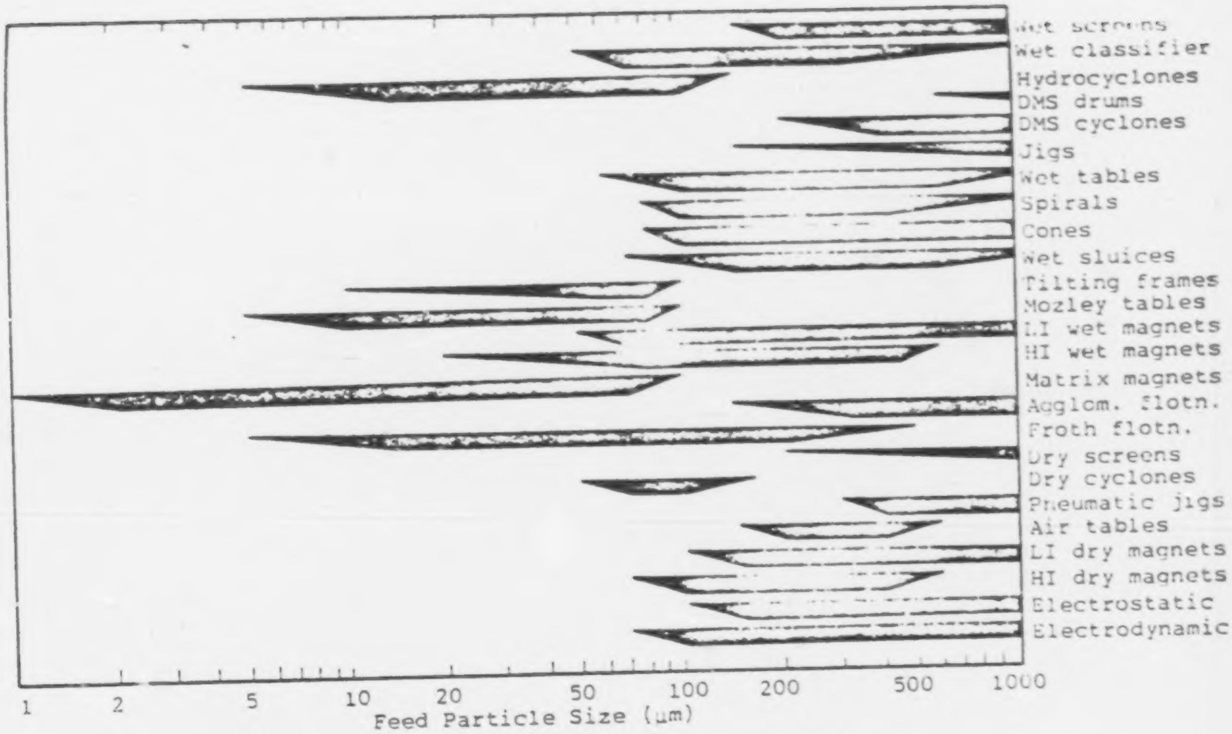
As discussed in Chapter I, there are seven disseminated chromite deposits in California that are of particular interest: Bar Rick, Pilliken, McGuffy Creek, Seiad Creek, Emma Bell, and North Elder Creek. All seven have some production history. The USBM Deposit Summary reports, as recently prepared by Nicholas Wetzel, suggest open-pit mining for Pilliken, McGuffy Creek, and North Elder Creek. Bar Rick ore could be recovered via sublevel stoping, and Seiad Creek/Emma Bell ore by a strip hillside method. Wetzel recommends gravity concentration of the chromite from all of these properties. He notes that the Cr_2O_3 content of the chromite grains in the disseminated chromite is quite variable, ranging from about 42% in the Pilliken deposit up to 48% Cr_2O_3 at Seiad Creek.

Work on beneficiation of low-grade California chromites was recently completed at the USBM Research Center in Salt Lake City (Salisbury, 1982). A combination of tabling and magnetic separation of a 6% Cr_2O_3 ore (Seiad Creek) yielded 36% Cr_2O_3 concentrate at 85% recovery. Ore from Auburn, California, was also treated. It contained only 2.5% Cr_2O_3, and required low-intensity magnetic separation in addition to gravity treatment to reject magnetite (Fe_3O_4). The gravity concentrate assayed 45% Cr_2O_3, accounted for 27% of the chromite, and had a Cr/Fe ratio of only 0.4. Overall recovery was thus 63%. A clean separation between magnetite and chromite was not obtained due to attachments of magnetite to the chromite grains (i.e., "liberation" was not complete). The investigators note that attempts to improve on these results via flotation "using known methods gave very poor results." Therefore, there is much still to be learned about concentrating these low-grade feedstocks.

Three key methods for upgrading domestic disseminated chromite are gravity, magnetic, and flotation separation. Gravity methods concentrate heavy minerals, including chromite and magnetite, and are effective for the coarser size fractions. Wet low-intensity magnetic separation removes magnetite, along with any chromite that is attached to grains of magnetite. Again, fine particles are not handled efficiently. Refer to Figure 6-1 (Mills, 1978). Final upgrading involves grinding the gravity and low-intensity magnetic concentrates sufficiently to separate the chromite mineralization from the unwanted minerals.

Two methods for concentrating fine particles of liberated chromite are high-intensity magnetic separation and flotation. Chromite is weakly magnetic, and responds to high-intensity treatment. An excellent example of use of those processes to concentrate chromite in the presence of magnetite is

FIGURE 6-1



Size Range Applicability of Commercial Unit Processes provided by T. Lukkarinen's description of processing off-grade (27% Cr_2O_3) ore from Kemi, Finland (300,000 TPY in 1973). The ore was ground to 40% minus 74-micron, from which the minus 40-micron slimes were removed. The slimes were treated by wet high-intensity magnetic separation. The coarse fraction was processed by gravity and low-intensity magnetics. Overall recovery was 70% into concentrates containing 42% tm 47% Cr_2O_3. The Cr_2O_3 content of the chromite mineral in the concentrate ranged form 39% to 51% Cr_2O_3.

The Finnish ore also responded well to flotation. The ore was crushed and ground to 100% minus 100-micorns, then deslimed at 35 microns. First, talc (magnesium silicate) was floated, followed by chromite flotation using an emulsion fuel oil and tall oil (a wood-pulp byproduct) at pH 6. The concentrate was then cleaned three times, the pH being lowered step by step with sulfuric acid (pH 5.0, 4.5, and 4.0, respectively). Overall recovery was 70% at 45% Cr_2O_3 grade. Tall oil, fuel oil, and acid consumptions were 2.6 lb/ton, 4.4 lb/ton, and 9 lb/ton, respectively. After completion of piloting, the Finnish metallurgists reportedly elected not to use flotation because of fear that the flotation tailing water could pollute the Gulf of Bothnia.

Chromite from Nickel-Cobalt Laterite

Domestic deposits of limonitic laterites, typically containing more chomium than Ni-Co, may at some time be mined for their Ni-Co values. Application of present Ni-Co leaching technologies (i.e., Moa Bay or Nicro) would leave most of the chromium in the leach residue as chromite. A chromite-recovery byproduct operation would not change the primary Ni-Co recovery plant. Coarse chromite may be recoverable via gravity treatment of the nickel plant feed. California Nickel reportedly could use gravity concentration to recover 100 ton/day of chromite assaying 40% Cr_2O_3 from

Gasquet Mountain nickeliferous laterite. Alternative access to the chromite would be via processing the nickel-leach residue. The residue chromium assay would be comparable to that of the original ore (i.e., about 2% Cr_2O_3). This alternative access is, however, a less desirable approach due to its apparent higher overall cost and lower grade chromite product. Physical beneficiation of the leach residue to yield a very low-grade chromite (15% to 20% Cr_2O_3 at 50% recovery) is feasible. The product could be suitable feedstock for the chromium chemicals industry, but would have a value of only a few cents per pound of contained chromium (Harvey, 1978). Recovery of chromium as a higher-value-added product, such as ferrochromium or sodium dichromate, would require a leach step to produce a high-Cr_2O_3 product. Capital and operating costs for this chemical upgrading would be high.

Further details on the technical and economic challenge of chromite recovery from domestic laterite sources were recently published by the USBM Albany Research Center (Kirby et al., 1982). In this work, leach residue generated by the Albany laboratory's roast-leach process for Ni-Co recovery from nickeliferous laterite (Siemens et al., 1977) was beneficiated using gravity-magnetic methods. Chromium recovery was 50% into a 33% Cr_2O_3 concentrate having a 1.6/1 ratio of Cr/Fe. "The product had neither the size distribution nor the composition preferred for metallurgical or refractory applications. Its best use would appear to be for manufacture of chromium chemicals." During the coming year, the USBM will be publishing results of an economic analysis of this process technology.

A rather innovative technique for using as-mined nickel laterite as a chromium source has been described by D. Slatter (1979). Ferrochromium ferronickel alloys (i.e., mixed alloys) were produced by electric smelting of composite charges of chromite and weathered nickeliferous serpentine (2% Ni, 1.5% Cr_2O_3). This approach was directed toward Rhodesia ore, in which these Cu-Ni sources occur in close association on the Great Dyke of Zimbabwe.

Production of Ferrochromium

Definite bright spots in advances toward utilization of domestic chromite reserves are recent developments in ferrochromium production technology. The importance of high Cr_2O_3 and Cr/Fe on economic incorporation of the chromium content of chromite into steel is thus decreasing. First, it is instructive to look at past efforts to utilize low-grade chromite concentrates as feedstock for ferrochrome production. The primary use of chromite is metallurgical, with traditional specifications for feedstock calling for at least 46% Cr_2O_3 and a Cr/Fe ratio of at least 2/1. An excellent review by R. Urguhart (1972) summarizes the numerous early efforts to utilize low-grade chromite for ferrochromium production.

Prereduction Followed by Leaching

Early work by the USBM involved roasting the chromite with carbon or methane, then leaching the selectively reduced iron with sulfuric acid. The agglomerated residue was smelted to high-carbon ferrochrome in a submerged-arc furnace. These efforts are described in USBM RI reports 3813, 3817, 3834, 3847, 4079, and 6010. Extensive testing failed to produce an economically attractive process (Urquhart, 1972). The salient problem was that the cost savings from producing low-grade concentrate were less than the subsequent roast-leach upgrading process.

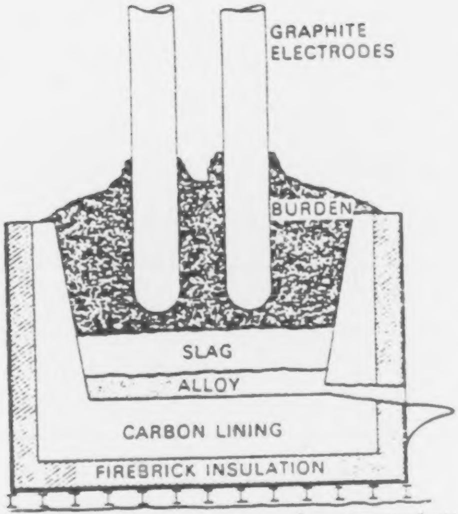
The use of thermochemical beneficiation to upgrade chromite was not limited to the USBM. Pinto et al. (1975) showed that roasting chromite for one hour at 900°C led to segregation of the Fe_2O_3 $2Cr_2O_3$ solid solution into discrete crystals, rendering the ore amenable to hydrometallurgical processing to remove iron. Robinson et al. (1981), applying a different approach, utilized reductive chlorination to remove iron from the chromite lattice. Off-grade chromite having a Cr/Fe ratio of 1.5/1 was converted to a product of 25/1 ratio. The Cr_2O_3 grade increased by one-third as a result of volatilization of iron chloride. Again, there is no indication that these roast-leach processes will become economic in the foreseeable future, particularly because of the decreasing importance of very high Cr/Fe ratios in chromite concentrates. Chemical processing to concentrate a heavy, inert, low price, weakly magnetic mineral such as chromite is, to this writer, inherently unattractive as compared to physical concentration.

Conventional Electric Furnace Operation

Whereas two decades ago the primary demand for ferrochromium was in the form of low-carbon alloy, high carbon is now the high-tonnage chromium alloy. This shift was the result of application of the AOD process (argon oxygen decarburization) for producing stainless steel. Carbon removal from the steel is carried out at reduced partial pressure of carbon monoxide to minimize chromium oxidation. The conventional furnace for manufacture of high-carbon ferrochromium is a three-phase submerged-arc furnace having three suspended carbon electrodes. The electrode tips are submerged under three to six feet of chromite ore, and are positioned above the slag layer (Figure 6-2, Urquhart, 1972). The open top of the furnace is continuously fed with chromite ore, a carbonaceous reducing agent, and the necessary fluxing agents.

During production of the ferrochromium, silica is partially reduced, as are the oxides of chromium and iron. About 60% of the phosphorous in the feed

FIGURE 6-2


GRAPHITE ELECTRODES
BURDEN
SLAG
ALLOY
CARBON LINING
FIREBRICK INSULATION

Schematic cross-section of a submerged-arc furnace used to produce high-carbon ferrochromium (the section is taken through two of the three electrodes) reports to the alloy. Sulfur in the charge is partly carried away with the furnace gases, and partly absorbed by the alloy. Furnace temperature is typically 1650°C to 1750°C, which assures slag and alloy fluidity and satisfactory reaction rates (Urquhart, 1972). An important facet of furnace operation is uniform distribution of ore burden to avoid channeling of furnace offgases and bridging. An increasing portion of western-world chromite is being produced as fines rather than lump ore. Blending fines with lump ore lowers the average ore price at the smelter. Fines lead to non-uniform distribution of ore burden, can plug ore-burden pores, and increase furnace dusting. Therefore, fines are being sintered, briquetted, or pelletized. Refer to the O'Shaughnessy paper (1982) for the flowsheets of the Cabo, Outokumpu, Showa Denko, and MS&A agglomeration processes. A further refinement of the pelletizing operation is the incorporation of a carbonaceous reductant. Some of the iron and chrome oxides are then reduced to the metal in a rotary kiln. The unit power consumption of the electric furnace is thus decreased (Sassos, 1982).

Two descriptions of ferrochromium production from domestic concentrates have been published recently by the USBM Twin Cities Research Center. Nafziger (Pittsburgh, 1980) has reviewed past USBM ferrochromium electric furnace efforts, and has provided a detailed description of prereduction of Stillwater chromite in a rotary kiln, followed by electric furnace smelting. Nafziger et al. (1980) also smelted a California disseminated chromite concentrate, and made estimates of energy, carbon, and slag requirements. Low-grade furnace feed escalates furnace energy consumption and slag volume. High slag volume leads to higher metal loss, because metal loss in the slag is proportional to slag volume. For a given ore and furnace, there is an optimum concentrate grade that is low enough to attain reasonable beneficiation costs, but not so low as to put undue burden on furnace costs. Where this ideal grade lies is a function of furnace and beneficiation technology, and on the distance between the concentrator and the furnace.

Recent Smelting Innovations

Two particularly significant innovations in commercial ferrochromium smelting technology have occurred recently: thermal-plasma smelting and the PMCO-T-LOY (PHA) process.

Thermal-Plasma Smelting

Foster Wheeler Energy Ltd. (England) developed a transferred-arc direct-current plasma gun that can be mounted on the roof of a ferrochromium furnace (Figure 6-3). It is processed by a hydraulic drive to distribute the energy over a large circular area (Monk, 1981).

Advantages of the plasma-furnace design illustrated in Figure 6-3 area:

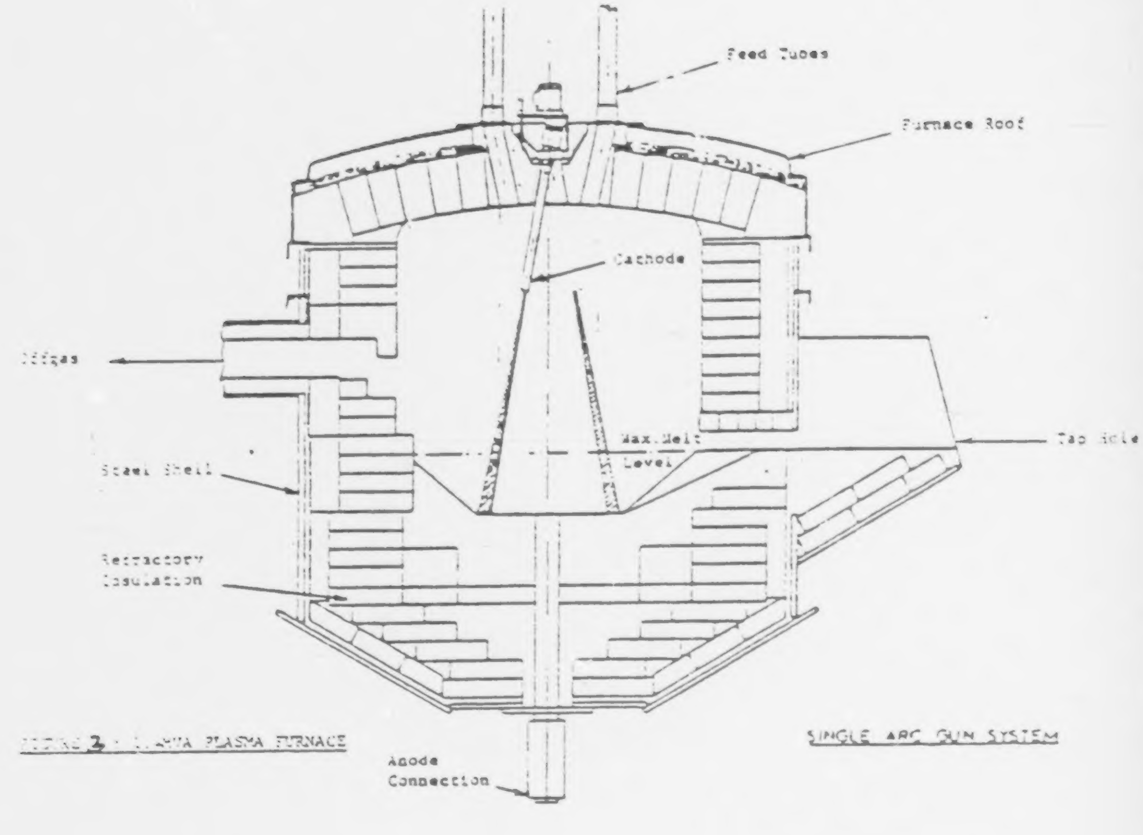
1. Agglomeration of the feed is not required. Coal, rather than metallurgical coke, is a suitable reductant. Need for carbon electrodes is eliminated.

2. Slag composition can be adjusted to maximize product recovery and maximize refractory wear, rather than to optimize energy input. Overall energy consumption is projected to be lower than for conventional processing. (Whether electrical energy consumption is also lower is not known to the writer.)

3. Reactions occur in a semi-closed system where gaseous environment can be controlled. Thermal losses are thus lowered. Maximum temperature occurs

FIGURE 6-3

PLASMA FURNACE


Feed Tubes
Furnace Roof
Cathode
Offgas
Max. Melt Level
Tap Hole
Steel Shell
Refractory Insulation
FIGURE 2 - [illegible] PLASMA FURNACE
Anode Connection
SINGLE ARC GUN SYSTEM

where it is most needed--at the slag-metal interface.

4. The high energy intensity of the plasma maximizes furnace throughput. Less than 10% of the rated power is lost in plasma-gun cooling and shell losses.

Middleburg Steel (Krugersdorp, South Africa) has announced plans to double its ferrochromium output via installation of a $25,000,000 thermal-plasma furnace (Preece, 1982). Start-up is planned for October, 1983. Work on development and application of thermal plasma for ferrochrome production began four years ago. It is claimed that installing the same capacity using conventional technology would cost $200,000,000. Middleburg's new 20-megawatt furnace is designed for an initial 25,000 tpy capacity, to be increased one year later to 50,000 tpy, presumably at added cost. One of the problems with smelting low-grade chromite concentrates, in addition to a high percentage of fines, is the high levels of impurities. Plasma smelting addresses this problem. During conventional electric smelting of chromite, volatilized sulfur tends to condense on the descending charge. This effect is minimized during plasma smelting, permitting production of a lower-sulfur product. With regard to phosphorous, plasma smelting operates with a higher temperature in the hearth zone. As a result, there is better removal of phosphorous via volatilization. It also turns out that plasma operation offers better control of the nitrogen content of the ferrochromium product (Pickles et al., 1978). Just how low grade a concentrate can economically be processed in a plasma furnace is not known to this writer.

Pamco Hot-Alloy Smelting

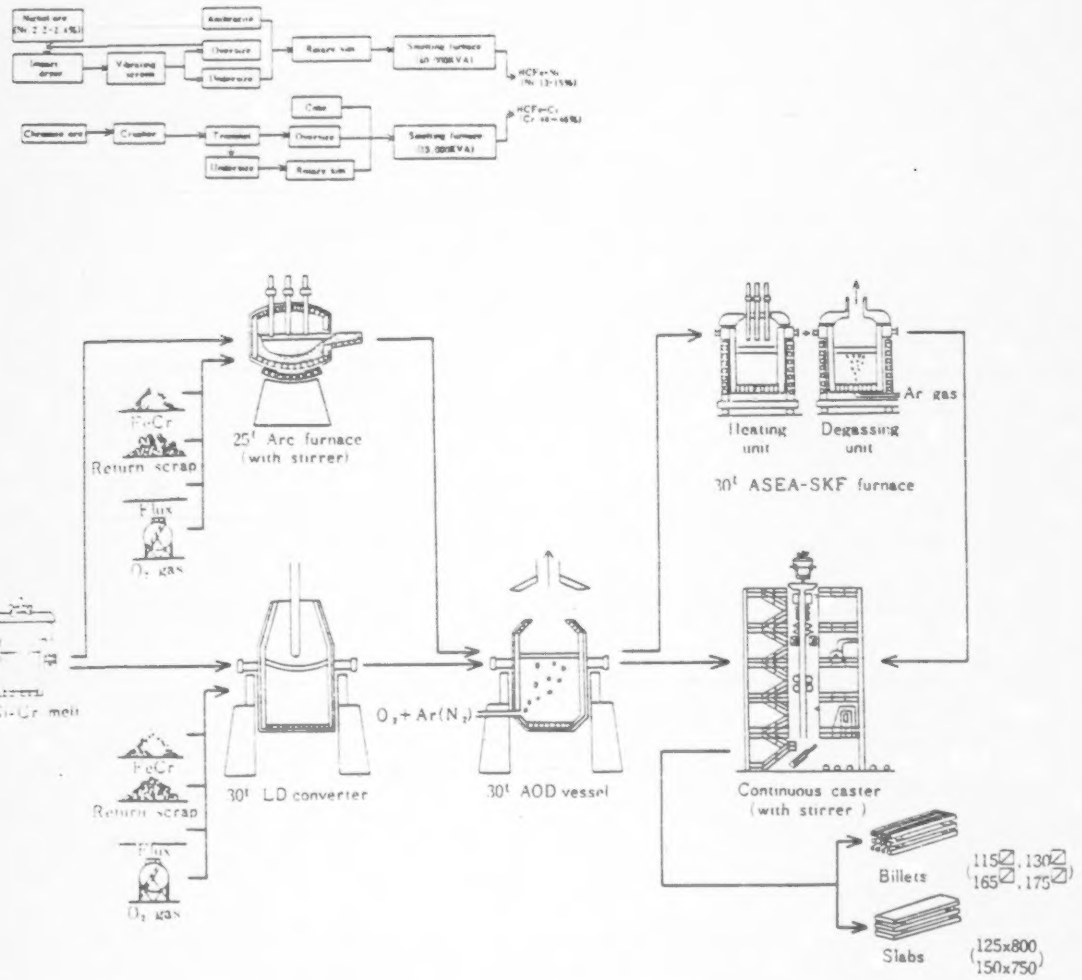
The second innovation, Pacific Metals' PHA process (Japan), is a direct method to produce stainless steel from low-grade feedstocks (Yamada et al., 1982). Nickel feed can be laterite ore containing 2.2% Ni. Chromite feed can be quite low in grade (40% Cr_2O_3 with a Cr/Fe of 1.5?). Total energy consumption is reduced by utilization of the manifest heat of molten Fe-Ni and Fe-Cr. Impurity levels of Sn, Sb, As, Cu, and Mo in the product steel are decreased because use of scrap, which contains these impurities, is avoided. The processing produces lower-than-normal levels of Ni (14%) in the Fe-Ni, and Cr (45%) in the Fe-Cr, due to iron dilution from the low-grade feed. Thus, the plant can dispense with the usual addition of steel scrap in the late stages of the stainless steel-making process (Figure 6-4). Note that 45% Cr in PHA ferrochromium is substantially lower than the 60% Cr produced by Nafziger (1980) from California low grade chromite. Thus, a traditional barrier to utilization of domestic chromite, its low Cr/Fe ratio and low Cr_2O_3 content, has been lowered.

COBALT

Introduction

The two most important potential domestic cobalt resources are Idaho's Blackbird Mine and mines in the Missouri Lead Belt. In this section, process technology for cobalt recovery from these sources is reviewed. Discussion of cobalt recovery from Missouri focuses on the Madison Mine, and on processing byproduct streams from existing lead mining operations. Start-up of copper-nickel mining operations from Minnesota's Duluth Gabbro would produce some cobalt. A fourth potential resource is byproduct cobalt from lateritic nickel deposits in the western United States. These byproduct sources are discussed in the section covering process technology for potential domestic nickel resources.

FIGURE 6-4


Nickel ore
Impact dryer
Vibrating screen
Anthracite
Oversize
Undersize
Rotary kiln
Smelting furnace
HCFe-Ni
(Ni 13-15%)
Chromite ore
Crusher
Trommel
Coke
Oversize
Undersize
Rotary kiln
Smelting furnace
HCFe-Cr
25t Arc furnace
(with stirrer)
FeCr
Return scrap
Flux
O2 gas
Fe-Ni-Cr melt
30t LD converter
FeCr
Return scrap
Flux
O2 gas
O2+Ar(N2)
30t AOD vessel
Heating unit
Degassing unit
Ar gas
30t ASEA-SKF furnace
Continuous caster
(with stirrer)
Billets
(115☐, 130☐, 165☐, 175☐)
Slabs
(125x800, 150x750)

Stainless steel production at the Hachinohe Works of the Pacific Metals Co. using the Pamco hot-alloy process.

The high prices of cobalt in the late 1970's resulted in substantial research efforts in recovery of this metal from domestic deposits. The proposed process at Blackbird would utilize flotation to produce a copper sulfide concentrate to be processed by conventional smelting, and a cobalt-arsenic sulfide concentrate containing cobaltitle (COAS?). The cobalt concentrate is treated under pressure to generate an impure aqueous cobalt liquor and a waste residue containing iron arsenate. The cobalt liquor would be purified, then neutralized to precipitate cobalt hydroxide, which would be used as feedstock for electrowinning of cobalt cathodes. The processing strategy for recovering cobalt from Anschutz Mining's Madison deposit, which is low in arsenic but high in lead, would be similar to that used for Blackbird. Flotation would be used to generate a galena (lead sulfide) concentrate for subsequent processing by smelting. A copper-cobalt-nickel sulfide concentrate would then be collected by flotation treatment of the solids rejected during lead flotation. The main Co-Ni mineral in the Cu-Co-Ni concentrate, siegenite, $(Co, Ni)_3S_4$, requires a sulfation roast to render the Co-Ni values soluble during the subsequent pressure leaching step. The Cu-Co-Ni liquor produced is treated for copper recovery, purified, then neutralized to produce a mixed Co-Ni precipitate that could be processed at AMAX's Port Nickel refinery or at a proposed smelter to be built by Anschutz at Fredericktown, Missouri. Ore from Missouri Lead Belt mines fed to lead sulfide flotation mills also contains siegenite, which becomes distributed between four mill outputs: tailings, $CuFeS_2$ concentrate, PbS concentrate, and ZnS concentrate. Each of these streams has potential for Co-Ni byproduct, again processed either by Anschutz or Post Nickel.

In Minnesota, flotation concentrates produced from Duluth Complex ore contain about 0.13% cobalt. There is also cobalt in western nickeliferous laterites that is potentially a significant byproduct. However, this cobalt almost certainly could not be exploited without significant government subsidy--substantially more subsidy than would be needed for the Blackbird or the Madison deposits. The western laterite deposits are small and low-grade, and not presently amenable to economic processing by proven processes. Discussion on recovery from these latter deposits is covered in Recovering Nickel from Domestic Deposits.

Conclusions

1. Process technology has been developed for recovering cobalt from the two most important domestic cobalt resources: Idaho's Blackbird deposit and Missouri's Lead Belt. Preliminary pilot-scale testing has been completed for Blackbird, for Missouri's Madison deposit, and for byproduct cobalt production from Missouri's lead mills. The flowsheets have not been piloted sufficiently to obtain the engineering data needed to design a commercial facility.

2. The Blackbird and Madison flowsheets provide bases for commercial operations, provided a high enough price for cobalt materializes (e.g., $20/lb.). A serious deficiency of the Blackbird process is the absence of a selenium-removal step. The Madison flowsheet could be simplified to produce crude Co-Ni sulfate crystals containing about 2 million annual pounds of cobalt for shipment to a central refinery, e.g., Port Nickel. A similar modification to Blackbird's flowsheet would enable production of impure cobalt-rich residue (sulfate, sulfide, hydroxide, or carbonate) containing selenium for shipment to Port Nickel for refining.

3. Approximately 0.5 to 1.5 million annual pounds of cobalt should be obtainable at reasonable cost as byproduct from Missouri Lead mill

operations. One opportunity involves production of a lower grade lead concentrate (e.g., 70% Pb). This modification would allow greater tonnages of Co-Ni sulfide to report to the byproduct copper concentrate, from which a Co-Ni concentrate could be prepared. A second opportunity involves production, from the mill tailings, of a pyrite concentrate containing nonferrous metals, including cobalt. This pyrite could be roasted, then chlorinated, to produce nonferrous metals, iron ore, and sulfuric acid.

4. AMAX's Port Nickel refinery in Louisiana could be upgraded to allow production of various cobalt and nickel products, including cathodes, briquettes, powder, and oxides. Feed sources could include Blackbird and Madison residues, Missouri Lead-mill byproduct, Duluth Gabbro matte, Ni-Co production from northwestern nickeliferous laterites, and open-market purchases of miscellaneous Ni-Co feedstocks. If such an approach were taken, no other cobalt refineries would need to be built to process ores from domestic deposits.

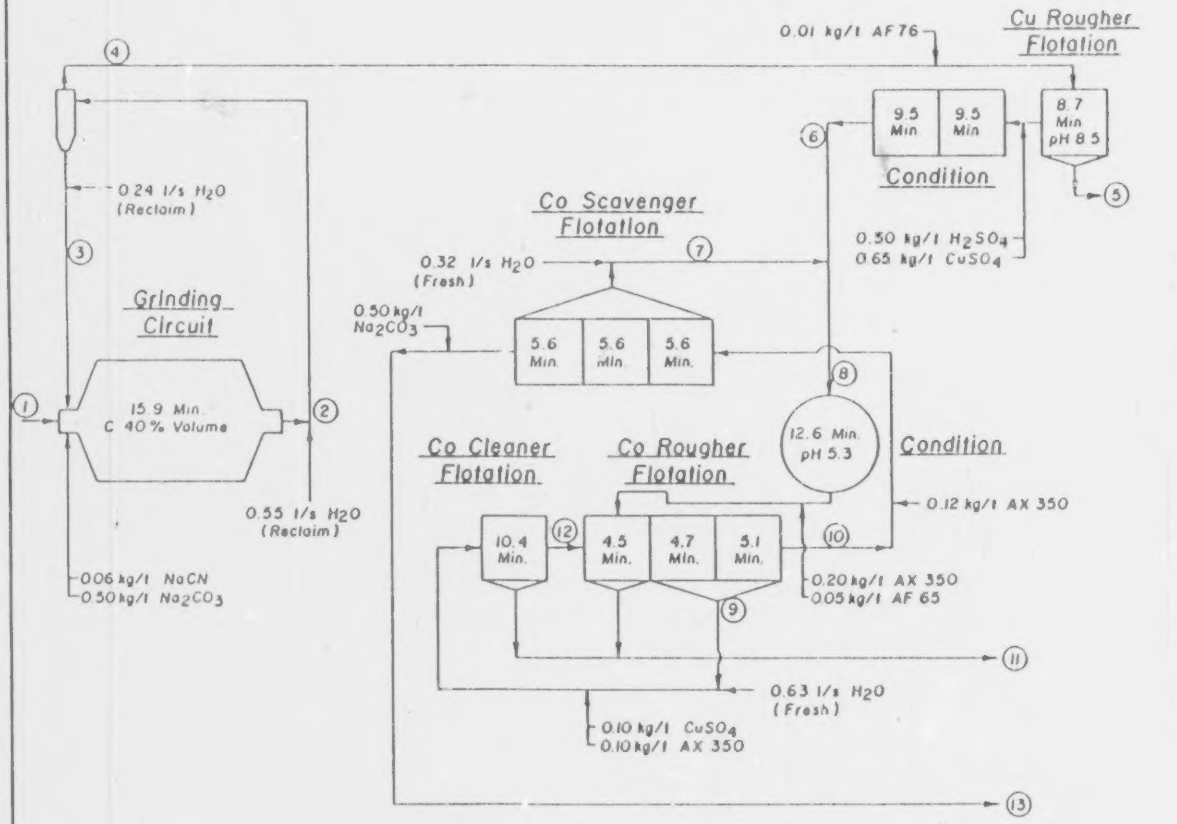
Idaho's Blackbird Mine

The first record of mining at Blackbird for recovery of cobalt dates from 1917 to 1920. Poor economics forced closure of the mine (Mitchell, 1982). Mine-development work was taken up by the Calera Mining Co. in 1943. Implementation of an extraction process proved far more difficult than had been anticipated, as will be explained below. Concentrate (17% Co, 24% As, 20% Fe, 29% S, 1% Ni, and 0.5% Cu) was shipped by rail to Garfield, Utah, for refining. Batches of ore were blended together to achieve an arsenic-to-iron weight ratio of 1.2, then pressure leached with sulfuric acid at 190°C and 500 psig using air. Residual iron and arsenic were precipitated by neutralizing the leach liquor, and copper was precipitated by cementation (i.e., replacement) with cobalt powder. Cobalt was recovered by hydrogen reduction during the first years of plant operation, and by electrowinning during the final year of operation. Prior to electrowinning, organics were extracted with benzene. Cobaltous hydroxide was then precipitated with lime (CaO), and used to feed the electrowinning circuit (Brooks, 1981). The refinery was operated intermittently from 1952 to 1959.

This operation proved to be a case study of equipment and process failure, which resulted in frequent interruptions in production (Coffer, 1953). Materials of construction, instrumentation, pump and valve design, cobalt powder recycle, autoclave scaling, agitator failure, filtration, and cobalt recovery-purity all combined to render Calera's refining efforts uneconomic (Martin, 1962). Final shutdown of the refinery in 1959 was reportedly caused by "poor economic conditions and completion of a Government Services Administration contract" (Mitchell, 1982). However, the inefficiency of the operation undoubtedly also played a role. Even if the Calera refinery had been well-engineered (Coffe's 1953 report shows that it was not), the primitive state-of-the-art of handling corrosive slurries at high pressure in the 1950's would probably still have led to frequent equipment failures.

Noranda Exploration, Inc., optioned the Blackbird Mine in 1977, and began site development in 1979. Laboratory and limited pilot-plant work were also undertaken to develop an improved beneficiation and refining flowsheet, as shown in Figures 6-5 and 6-6. The refining flowsheet was tested on a continuous basis using 10-kg lots of concentrate per cycle for 55 cycles. Sufficient design data were collected for future construction of a pilot plant, operation of which will be necessary to obtain engineering design data

Blackbird Pilot Plant


0.01 kg/t AF76
Cu Rougher Flotation
8.7 Min pH 8.5
9.5 Min
9.5 Min
Condition
0.50 kg/t H2SO4
0.65 kg/t CuSO4
0.24 l/s H2O (Reclaim)
Grinding Circuit
15.9 Min. C 40% Volume
0.55 l/s H2O (Reclaim)
0.06 kg/t NaCN
0.50 kg/t Na2CO3
Co Scavenger Flotation
0.32 l/s H2O (Fresh)
0.50 kg/t Na2CO3
5.6 Min.
5.6 Min.
5.6 Min.
12.6 Min. pH 5.3
Condition
0.12 kg/t AX 350
Co Cleaner Flotation
10.4 Min.
Co Rougher Flotation
4.5 Min.
4.7 Min.
5.1 Min.
0.20 kg/t AX 350
0.05 kg/t AF 65
0.63 l/s H2O (Fresh)
0.10 kg/t CuSO4
0.10 kg/t AX 350
①
②
③
④
⑤
⑥
⑦
⑧
⑨
⑩
⑪
⑫
⑬

	Tonne/h % Sol	m³/h l/s	% Co % Cu	
①	2.72 97.5	1.04 0.29	0.61 1.08	Mill Feed
②	7.21 62.00	7.02 1.96	1.49 ——	Cyclone Feed
③	4.49 65.6	3.99 1.11	2.02 ——	Cyclone U' Flow
④	2.72 48.0	3.92 1.10	0.61 1.08	Cyclone O' Flow
⑤	0.08 20.00	0.36 0.10	0.62 28.49	Cu Rough Conc
⑥	2.64 50.30	3.57 1.00	0.61 0.26	Cu Rough Tail
⑦	0.06 4.21	1.49 0.41	2.30 1.80	Co Scav Conc
⑧	2.70 40.00	5.06 1.40	0.65 0.29	Co Circuit Feed
⑨	0.23 6.70	3.26 0.90	4.00 1.60	Co Rough Conc
⑩	2.42 31.80	6.08 1.69	0.13 0.09	Co Rough Tail
⑪	0.28 20.00	1.19 0.33	5.17 2.05	Co Conc
⑫	0.06 2.50	2.56 0.72	2.50 1.50	Co Clnr Tail
⑬	2.36 32.40	5.71 1.61	0.063 0.044	Final Plant Tail

Pilot Plant
10/81 NTS
BLACKBIRD
noranda
Noranda Mining Incorporated

FIGURE 6-5

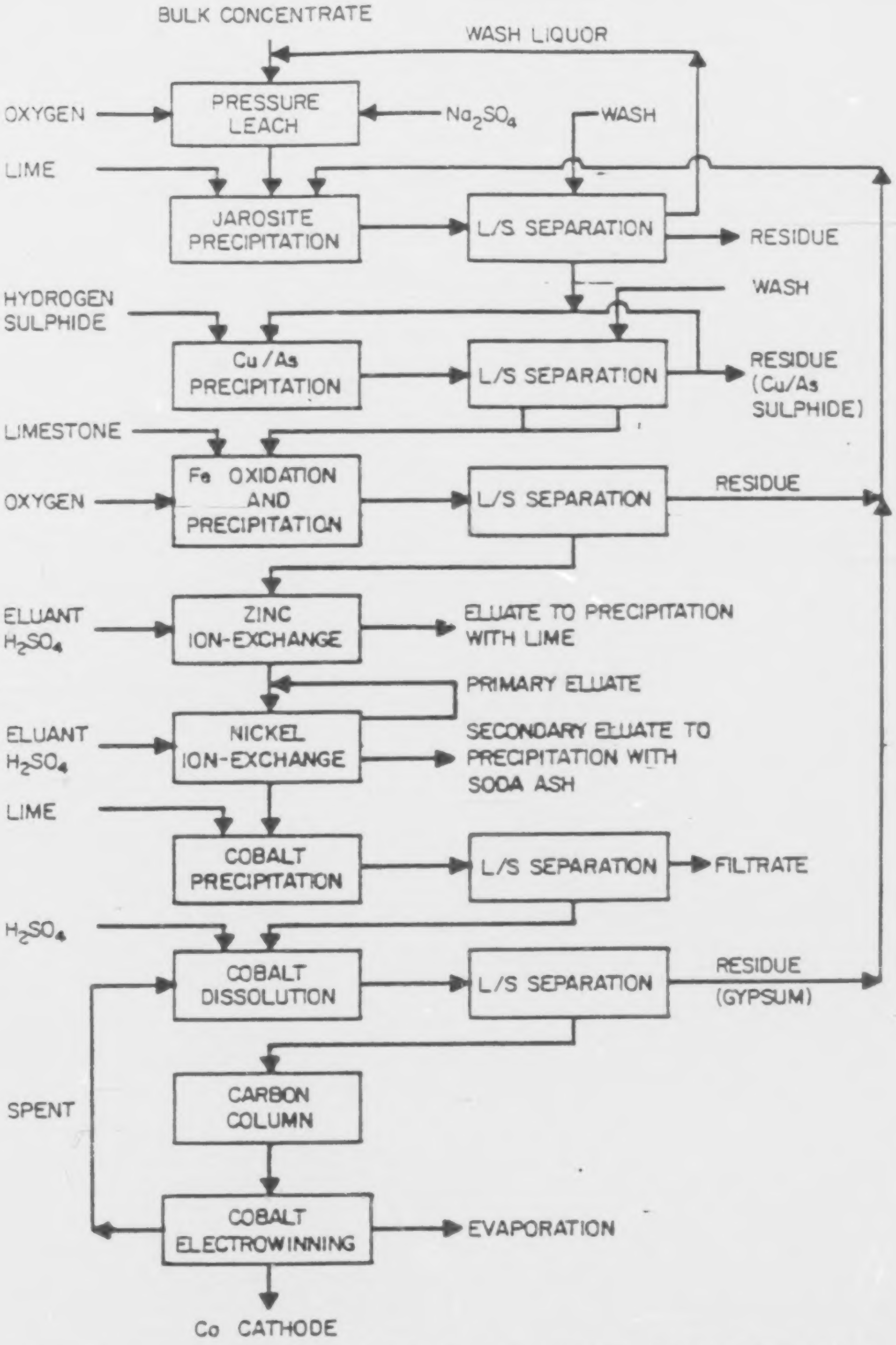

BULK CONCENTRATE
WASH LIQUOR
OXYGEN
PRESSURE LEACH
Na_2SO_4
WASH
LIME
JAROSITE PRECIPITATION
L/S SEPARATION
RESIDUE
WASH
HYDROGEN SULPHIDE
Cu/As PRECIPITATION
L/S SEPARATION
RESIDUE (Cu/As SULPHIDE)
LIMESTONE
OXYGEN
Fe OXIDATION AND PRECIPITATION
L/S SEPARATION
RESIDUE
ELUANT H_2SO_4
ZINC ION-EXCHANGE
ELUATE TO PRECIPITATION WITH LIME
PRIMARY ELUATE
ELUANT H_2SO_4
NICKEL ION-EXCHANGE
SECONDARY ELUATE TO PRECIPITATION WITH SODA ASH
LIME
COBALT PRECIPITATION
L/S SEPARATION
FILTRATE
H_2SO_4
COBALT DISSOLUTION
L/S SEPARATION
RESIDUE (GYPSUM)
SPENT
CARBON COLUMN
COBALT ELECTROWINNING
EVAPORATION
Co CATHODE

FIGURE 6-6 Flow Diagram for the Treatment of Blackbird Bulk Concentrate.

for the commercial facility.

The old Calera mill floated chalcopyrite, while depressing (sinking) cobaltite (CoAsS) and Pyrite (FeS2), at pH 10.4 and $35^{o}C$ to $40^{o}C$. The ore assayed approximately 0.7% Co, 1.6% Cu, plus traces of nickel. Cobaltite was activated (floated) with sodium sulfide at pH 3.2, reportedly rejecting (sinking) an 0.06% Co tailing (Douglas, 1956). The actual tailing analysis was probably 0.13%, with 75% of the ore's cobalt content reporting to the cobalt concentrate (Mitchell, 1982). A typical Calera concentrate contained cobaltite and pyrite, with varying small percentages of gersdoffite (NiAsS), chalcopyrite, and arsenopyrite (FeAsS). The gangue minerals were mica, $KAl_2Si_3O_{10}(OH)_2$, quartz, SiO_2, and chlorite, $H_8Mg_5Al_2Si_3O_{18}$.

With this background, Noranda assembled three flotation flowsheets, and performed cost evaluations in each. The flowsheet selected is shown in Figure 6-5 (Mitchell, 1982). It differs from the Calera mill operation primarily in its emphasis on attaining higher cobalt and copper recovery, while accepting a lower grade cobalt concentrate. Mill feed assaying 0.61% Co and 1.1% Cu was treated by flotation to generate a high-grade copper concentrate (28% Cu) by using starvation quantities of a weak collector (pH 8.5), which maximized the grade of the copper concentrate while depressing cobalt. Copper recovery was 78%. Only 3% of the cobalt reported with the copper concentrate. The tailings from copper flotation were then conditioned and floated, as shown in Figure 6-5 to produce cobalt concentrate assaying 5% Co and 2% Cu. The tailings from this second flotation step contained 0.06% Co. Thus, cobalt recovery into the cobalt concentrate was 88%, with the remaining cobalt lost to the mill tailings (9%) and to the copper concentrate (3%).

For size reduction of the as-mined ore, Noranda has elected to expand the existing mill to 1200 tpd, assuming the project goes forward, in the following manner:

> Run-of-mine ore from the Blackbird Mine would be crushed to about 5/8-inch in a primary crushing circuit located at the 6850-level portal. The existing components, consisting of one jaw crusher and one screen and one cone crusher, have a capacity of 100 tons per hour, which would be adequate to supply the expanded concentrator. The run-of-mine ore would be transported from the coarse ore storage to the primary crushing circuit by means of a pan-feeder. Crushed ore would be transported from the crusher to a new fine ore bin, to be located adjacent to the concentrator, by means of the existing 22-inch covered belt conveyor. The existing conveyor has a capacity of 150 to 500 TPH which would not have to be increased for the 1200 tpd operation. (Salmon National Forest 1982).

A grinding and flotation material balance published by Noranda shows three ball mills for grinding the crushed ore (Mitchell, 1982). Two of the mills are 8 feet diameter by 9 feet long, the third is 7 feet by 11 feet. The mills would be operated in single stage in closed circuit with cyclones, which separate the ground ore into coarse and fine fractions. Copper flotation feed is to be 80% minus 90 microns.

Cobalt Concentrate Refining

The cobalt concentrate produced by Noranda's milling flowsheet assays 5% to 7% Co, 1% to 2% Cu, 8% to 13% As, 25% to 35% Fe, and 30% to 40% S, 5% to 10% SiO_2, 0.2% Ni, plus traces of heavy metals such as Zn. Cobaltite and arsenopyrite are the primary cobalt-bearing minerals. The refining flowsheet developed by Noranda is shown in Figure 6-6 (Harris 1982). The concentrate is pressure leached in sulfuric acid at 150°C, 175 psi oxygen, 30% solids, and 30 g/l Na_2SO_4. The bulk of the arsenic precipitates during leaching as ferric arsenate. Addition of sodium sulfate accelerates the precipitation of iron as jarosite. The cobalt-rich liquor (10 gpl H_2SO_4) is treated as follows:

1. Neutralization of the pressure leach slurry to pH 1.5 at 95°C with recycled iron oxidation cake, recycled gypsum cake ($CaSO_4$) from cobaltous hydroxide dissolution in spent electrolyte, and 20% lime (CaO) slurry to complete precipitation of iron and arsenic as jarosite, $3Fe_2O_3\ 4SO_3\ 9H_2O$, and ferric arsenate;

2. H_2S treatment of the primary jarosite filtrate at its natural pH at 45^o to 60^oC, with 50% recycle of the sulfide residue to act as seed for copper and arsenic removal, followed by oxidation and precipitation of residual iron at pH 5.0 and 75^oC using 40% limestone slurry and oxygen;

3. Removal of zinc with the Bayer ion exchange resin, Lewatit OC-1026 at pH 4.0 and ambient temperature (20^o to 25^oC); removal of nickel with the Dow ion exchange resin, XF-4195, at pH 2.5 and ambient temperature (Rosato, 1982);

4. Precipitation of cobalt as cobaltous hydroxide with 20% lime slurry at pH 8.5 and ambient temperature; dissolution of cobaltous hydroxide with spent electrolyte to pH 6.5; and followed by electrowinning of cobalt cathodes using proprietary anodes and cathode blanks, at a current density of 20 to 26 $amps/ft^2$.

Overall recovery of cobalt from the concentrate was 95%, with most of the losses being to the leach residue-jarosite precipitate mixture. For each ton of concentrate processed, about 1.8 tons of residue-jarosite waste was produced assaying approximately 17% Fe and 6% As. Sulfide precipitation of residual copper and arsenic from the cobalt liquor (i.e., the "pregnant" liquor) generates 0.1 tons of cake assaying about 22% Cu, 11% As, 6% Co, and 0.3% Ni. This residue contains about 1% of the cobalt in the feed concentrate. It could probably be sold to a copper smelter, including two operated by Noranda in Quebec.

The Blackbird flowsheet developed by Noranda utilizes unit operations that have been proven commercially, with two exceptions. One is the nickel ion-exchange and the other is the zinc-ion exchange step. Both appear to be commercially viable. The leach residue has met requirements for non-hazardous waste based on standard EPA tests. The most serious problem with this flowsheet is the absence of a selenium-removal step. The cobalt cathodes contain about 10 ppm Se, which exceeds the selenium content of commercial cathodes by a factor of ten. Noranda engineers believed that they were producing low-selenium cathodes during their process development program. Only during the final months of their testing did they learn of faulty selenium assays. At this date, Noranda has not found a cost-efficient method to reject selenium. Noranda's overall estimated costs for producing cobalt in Idaho, $20.00 to $25.00 per pound Co, have seemed reasonable; and four years should be sufficient to complete piloting, engineering, and plant construction. However, this cost and timing does not include removing selenium. Low selenium content is essential for many end uses of cobalt. Selenium-removal technology, such as vacuum-refining the cathodes, is available, but is expensive. A logical alternative is to ship the selenium-contaminated cobalt hydroxide to a central refinery, such as Port Nickel. Port Nickel is presently not equipped to produce low-selenium cobalt, a deficiency that could (and should) be corrected. Thus, Noranda has yet to successfully confront all refinery problems.

An alternative method to process arsenical cobalt concentrates is that used by Penarroya for Moroccan concentrates (Lenoir, 1981). Their concentrate is roasted at about 650 °C to volatilize arsenic trioxide, which is collected as fume and sold. The calcine (i.e., the roasted solids) is leached with aqueous $HCl-Cl_2$. The pregnant liquor is purified, then electrolyzed to produce cobalt cathodes. The Metaux Speciaux plant at Plombieres, France, produces 1500 tpy cobalt in this manner. Their cathodes presumably meet commercial specifications. Noranda chose not to use a process involving roasting "due to environmental constraints" (Harris, 1982). Whether these constraintsinvolved airborne arsenic, or control of aqueous chloride, or other factors, is not known. In view of the selenium problem in Noranda's autoclave-based flowsheet, the Penarroya process deserves closer examination.

The Madison Mine, Missouri

The Madison deposit is located at Fredericktown, Missouri, Anschutz has rehabilitated portions of the mine. During the 1950's, National Lead built and operated a mine, mill, and refinery for recovering lead and copper concentrate, and cobalt and nickel powder (Lutjen, 1953) but the mill and refinery are gone. The National Lead refinery process involved ammoniacal pressure leaching of concentrate containing about 7% Co as siegenite. This mineral is exceptionally stable, and requires high-tempertature autoclave treatment in order to oxidize the sulfur and dissolve the Co-Ni values (Bilbrey, 1962). The process failed for many of the same reasons Calera's operation at Garfield failed: the state-of-the-art of high-pressure process equipment for handling corrosive slurries was not sufficiently advanced at the time.

Anschutz Corp. (Denver) bought the Madison Mine claims in 1979, and has since been planning a Co-Ni-recovery operation based on a 10-year mine life. Anschutz reportedly plans an approximately 2000 tpd operation, followed by crushing, grinding, and flotation production of a lead and a copper-nickel-cobalt concentrate (Figure 6-7). Mill feed for underground ore is to be 0.22% Co, 0.27% Ni, 0.7% Pb, and 0.5% Pb. Projected metal recoveries from ore to concentrate are 95+% for copper, and 85% each for nickel and cobalt. Target cobalt production would be 2 million lb/yr (Amaro, 1980).

Old mill tailings are to be processed according to the block diagram given in Figure 6-8 (Amaro, 1980). Anschutz hopes to process sufficient tailings containing 0.1% Co, 0.07% Ni, 4% Fe, 0.06% Cu, 0.05% Pb, and traces of arsenic to produce approximately 200 tpd of concentrate assaying 2.3% Co, 1.6% Ni, 42% Fe plus traces of Pb and Cu. Oxide coatings on these tailings would first be broken up by crushing, followed by two stages of grinding. Anschutz expects that nickel and cobalt recovery from the tailings into the concentrate will be 65%. This lower recovery is due both to the low grade of the feed, and to the oxide coatings. Tailings mine life would be three years, with a target production of 2 million lb Co/yr.

Co-Ni Concentrate Refining

Details of the Anschutz process for refining the Co-Ni concentrate have recently been published by Sherritt Gordon Mines, developer of the process (Weir, 1982). The concentrate is first given a sulfation roast, then leached, as shown in Figures 6-9, -10, and -11). This flowsheet was piloted at Sherritt Gordon's Ft. Saskatchewan facility, processing 400 lb per day of calcine assaying 1.7% Co, 2.3% Ni, 5.9% Cu, 40% Fe, 0.2% Zn, 0.2% Mn, 0.9% Pb, and 8% S. Leaching conditions are much less severe than those in the old National Lead operation. The Anschutz pressure leach operates at 150°C (vs 230°C), 50 psi oxygen partial pressure, 1 hr, without sulfuric acid addition. These conditions are comparable to modern nickel-matte leaching operations, which function smoothly. The lower temperature operation is

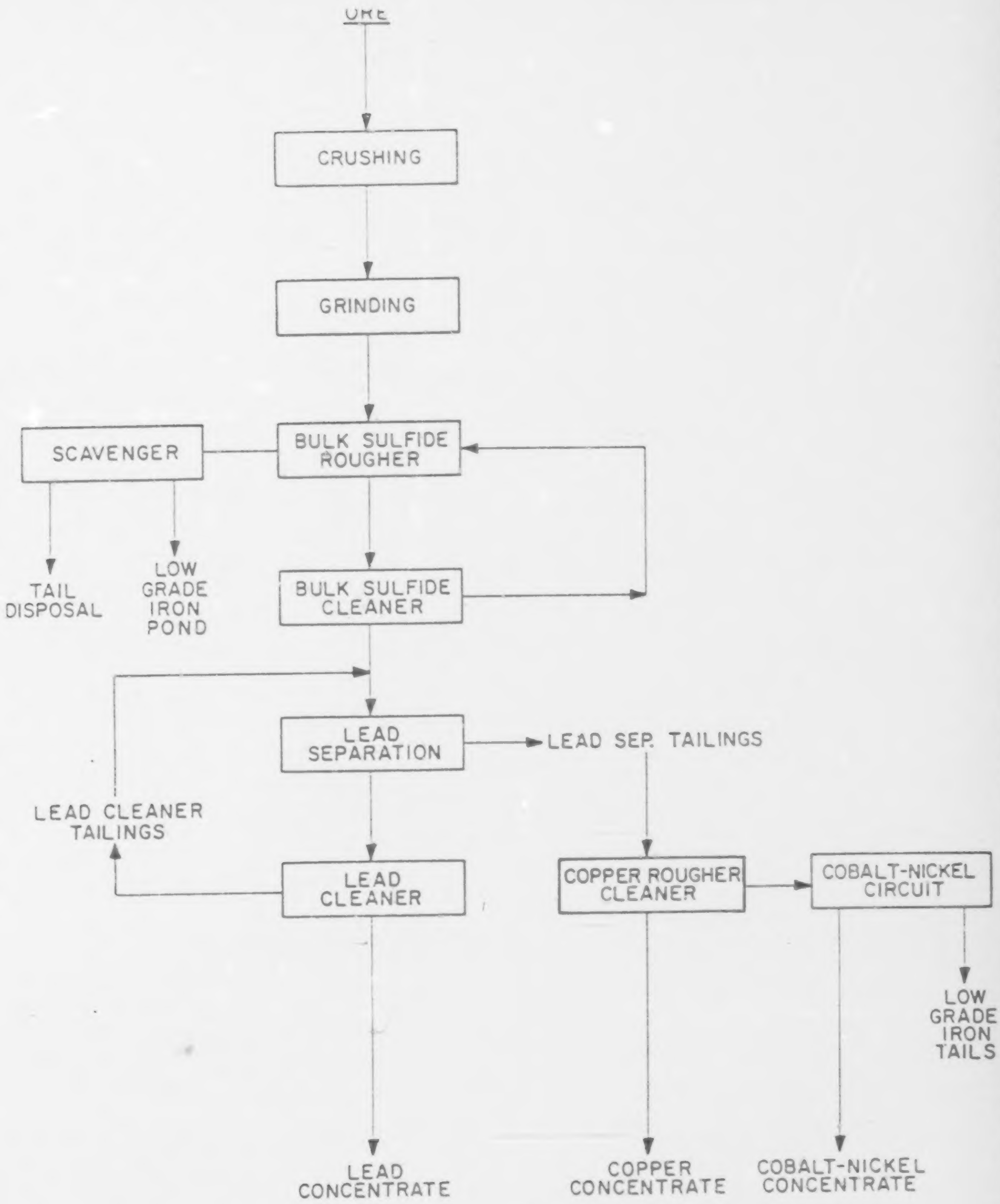

ORE
CRUSHING
GRINDING
SCAVENGER
BULK SULFIDE ROUGHER
TAIL DISPOSAL
LOW GRADE IRON POND
BULK SULFIDE CLEANER
LEAD SEPARATION
LEAD SEP. TAILINGS
LEAD CLEANER TAILINGS
LEAD CLEANER
COPPER ROUGHER CLEANER
COBALT-NICKEL CIRCUIT
LOW GRADE IRON TAILS
LEAD CONCENTRATE
COPPER CONCENTRATE
COBALT-NICKEL CONCENTRATE

FIGURE 6-7 Flow Diagram of Anschutz's Proposed Mine-Run Ore Beneficiation Plant. The refinery would process the combined copper and cobalt-nickel concentrate.

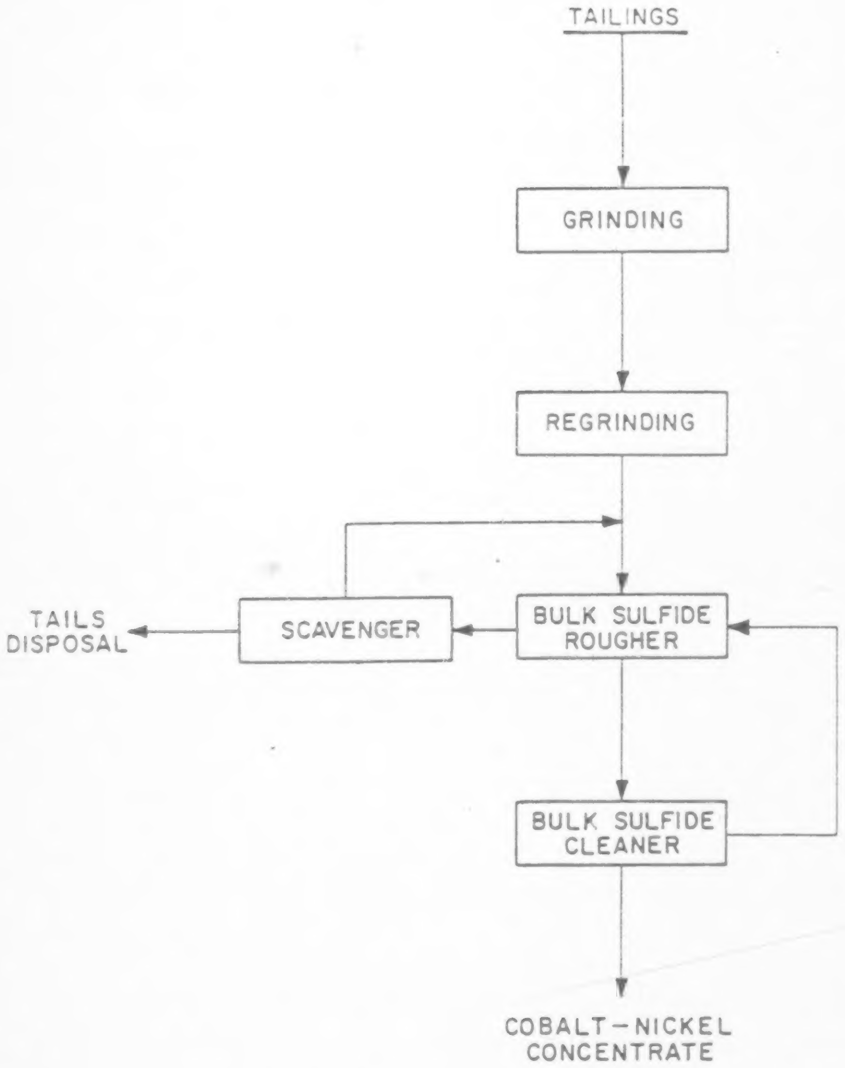
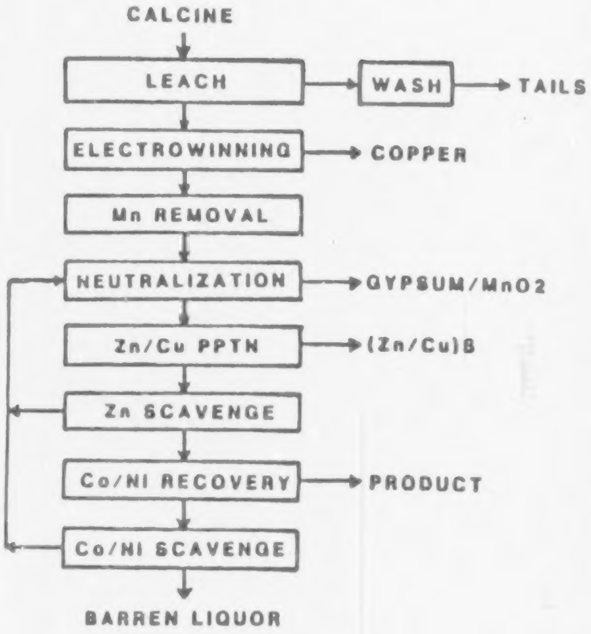

TAILINGS
GRINDING
REGRINDING
SCAVENGER
BULK SULFIDE ROUGHER
TAILS DISPOSAL
BULK SULFIDE CLEANER
COBALT-NICKEL CONCENTRATE

FIGURE 6-8 Flow Diagram for Anschutz's Proposed Tailing-Ore Beneficiation System

METALLURGICAL

BALANCE

FIGURE 6-9


CALCINE
LEACH
WASH
TAILS
ELECTROWINNING
COPPER
Mn REMOVAL
NEUTRALIZATION
GYPSUM/MnO2
Zn/Cu PPTN
(Zn/Cu)S
Zn SCAVENGE
Co/Ni RECOVERY
PRODUCT
Co/Ni SCAVENGE
BARREN LIQUOR

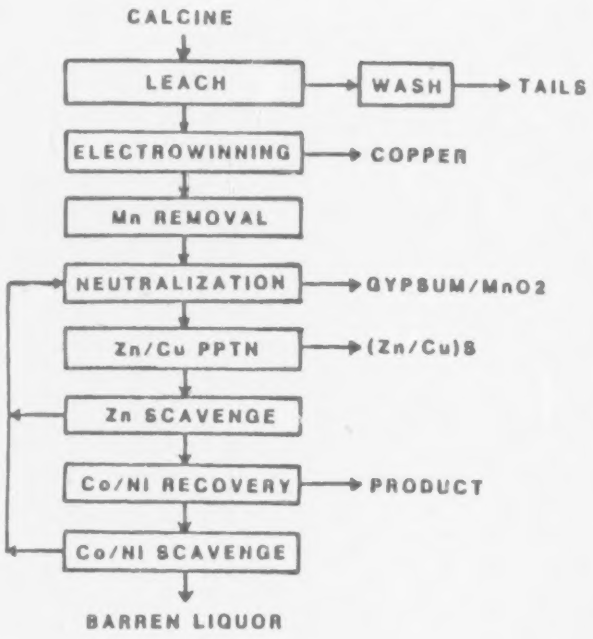
Assumes no inventory changes

COBALT	FLOW g,L	ANAL. %,g/L	DIST. %
Calcine	1000	1.78	100
Tails	690	0.150	5.8
Tails	0.72	0.065	0.3
Cathode	24.7	0.0003	–
Powder	33.7	<0.001	–
Gypsum	177	0.06	0.6
(Zn/Cu)S	12.5	0.7	0.5
Product	74.1	20.1	83.7
Barren	2.72	0.02	0.3
Un-accounted	–	–	8.8

METALLURGICAL

BALANCE

FIGURE 6-10

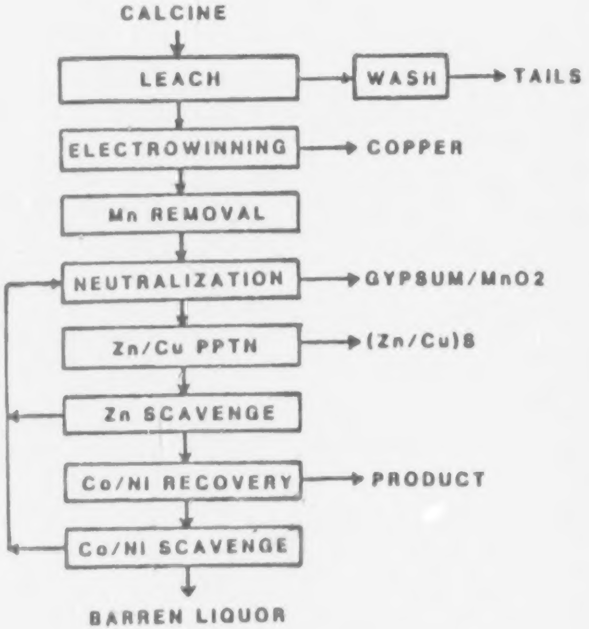

CALCINE
LEACH
WASH
TAILS
ELECTROWINNING
COPPER
Mn REMOVAL
NEUTRALIZATION
GYPSUM/MnO2
Zn/Cu PPTN
(Zn/Cu)S
Zn SCAVENGE
Co/Ni RECOVERY
PRODUCT
Co/Ni SCAVENGE
BARREN LIQUOR

Assumes no inventory changes

NICKEL	FLOW g,L	ANAL. %,g/L	DIST. %
Calcine	1000	2.40	100
Tails	690	0.448	12.9
Tails	0.72	0.083	0.2
Cathode	24.7	<0.001	-
Powder	33.7	0.003	-
Gypsum	177	0.05	0.4
(Zn/Cu)S	12.5	0.78	0.4
Product	74.1	24.6	76.0
Barren	2.72	0.04	0.5
Un-accounted	-	-	9.6

METALLURGICAL BALANCE

FIGURE 6-11


CALCINE
LEACH
WASH
TAILS
ELECTROWINNING
COPPER
Mn REMOVAL
NEUTRALIZATION
GYPSUM/MnO2
Zn/Cu PPTN
(Zn/Cu)S
Zn SCAVENGE
Co/Ni RECOVERY
PRODUCT
Co/Ni SCAVENGE
BARREN LIQUOR

Assumes no inventory changes

COPPER	FLOW g,L	ANAL. %,g/L	DIST. %
Calcine	1000	5.94	100
Tails	690	0.442	5.1
Tails	0.72	0.136	0.2
Cathode	24.7	>99	41.0
Powder	33.7	87	49.0
Gypsum	177	0.02	-
(Zn/Cu)S	12.5	3.29	0.7
Product	74.1	0.003	-
Barren	2.72	0.001	-
Un-accounted	-	-	3.9

feasible due to prior sulfation roasting of the concentrate, which decomposes the Co-Ni sulfide, principally into more soluble Co-Ni sulfates. Sherritt Gordon engineers had hoped that the calcine could be leached atmospherically. Atmospheric treatment gave satisfactory results for cobalt but left 30% of the nickel in the leach residue. Note that pressure-leaching also allows more complete iron rejection, resulting in a readily filterable leach residue.

The autoclave step, as piloted by Sherritt Gordon, discharges about 20 gpl (grams per liter) H_2SO_4. Original development of the process was carried out at Hazen Research, Inc., Golden. This relatively low acidity is an excellent medium for the iron oxides precipitated during leaching to scavenge selenium from solution. Note that some selenium has already been fumed from the concentrate during the prior roasting step. What selenium does make it into the pregnant liquor is removed during the subsequent copper electrolysis, mainly during the high-current-density portion of the operation. In contrast, Noranda's flowsheet has no roasting step prior to leaching. Their autoclave discharge liquor contains 100 gpl H_2SO_4, an ideal medium to precipitate ferric arsenate, but poor for selenium coprecipitation. Furthermore, the Noranda flowsheet does not incorporate copper electrowinning. Thus, all three selenium removal steps present in the Anschutz flowsheet are absent in the Noranda flowsheet.

The pregnant liquor generated by autoclaving the calcine contains about 10 g/l Co, 12 g/l Ni, 30 g/l Cu, and 20 g/l H_2SO_4. Copper is electrowon in two stages, the first producing copper cathodes, and the second copper powder. Copper cathodes are produced from the copper-rich liquor (30 g/l Cu down to 18 g/l Cu) and powder from the copper-poor liquor (18 g/l Cu down to 1 g/l Cu). The weight ratio of cathode to powder product is about one. The cathodes contain up to 50 ppm Pb which would probably necessitate subsequent electrolytic refining. The copper powder (2500 ppm Pb) could be shipped to a copper smelter. Alternatively, it could be melted, cast into anodes, and electrorefined.

After recovering copper, manganese is precipitated by oxidizing soluble Mn^{+2} to insoluble Mn^{+4} using ozone gas. Aqueous manganese is lowered from 500 ppm down to trace levels. The product slurry (70 g/l H_2SO_4) is then neutralized to pH 3 using limestone ($CaCO_3$). The waste residue is gypsum ($CaSO_4$) containing 25% Ca, 1% Fe, 0.5% Mn, 0.05% Co, 0.02% Ni, plus traces of heavy metals. The next process step is the removal of residual copper. Copper and a portion of the zinc (1.3 g/l) are precipitated as sulfides by addition of H_2S gas to the neutralized solution. Solution pH is maintained at about 5.5 via addition of limestone. The product liquor assays 4 ppm Cu and 0.7 g/l Zn. The sulfide precipitate contains 12% Ca, 9% Cu, 1.6% Ni, 0.7% Co, and 8% Zn. This cake, which provides an exit for zinc, would most likely be blended with the gypsum cake and sent to waste. Final zinc removal from the filtered H_2S-treated solution is done by further neutralizing to pH 6.5 with calcium hydroxide to generate a solution containing 10 g/l Ni, 8 Co, and 20 ppm Zn. The residue, which contains significant Ni-Co values, is recycled back to the primary acid neutralization operation (see Figure 6-9). Here the residue is blended with the acidic slurry discharged from the ozonation step. The residue's metal values dissolve in the acid.

A mixed Ni-Co carbonate is precipitated from the pH 6.5 low-Zn liquor by addition of soda ash (i.e., Na_2CO_3). Analysis of the mixed salt (wt %) is tabulated below:

Element	Wt %	Element	Wt %
Ni	27.0	Cu	0.01 %
Co	22.0	Fe	0.005
SO_4	3.0	Zn	0.006
Ca	0.3	Se	0.001
Na	0.1	Pb	0.01
Mg	0.1	Cd	0.001
Mn	0.1	Sb	0.001

The pH 6.5 liquor effluent from the Ni-Co carbonate precipitation step contains 0.3 g/l Ni and 0.2 g/l Co. It is neutralized with soda ash to pH 8.5, to precipitate residual Ni-Co values. The barren liquor, assaying 25 ppm Ni and 20 ppm Co, is sent to waste treatment. The impure Ni-Co carbonate precipitate is recycled back to the primary acid neutralization step (Figure 6-9). Anschutz engineers have subjected their mill and refinery wastes to the standard EPA tests, and found them to be non-hazardous in terms of regulatory standards. The state of Missouri is reportedly not certain that the federal standards are sufficiently strict (Daniels, 1982).

The Ni-Co carbonate product described in the Weir paper is probably not ideal for many end uses. A less pure Ni-Co sulfate is an alternative product. Dr. R. Weir has suggested the following alternative approach (pers. comm.). The concentrate could be leached with dilute acid generated from roaster off-gases to selectively solubilize and thus remove most of the magnesium and manganese prior to roasting, analogous to the standard pretreatment often used for zinc concentrates. Following autoclaving, one could then neutralize the product liquor from copper electrowinning to pH 4.5, then precipitate a mixed Ni-Co sulfate by evaporative crystallization. Similar technology is standard for removing nickel from copper-tankhouse bleed liquor. A portion of the energy required for this evaporation could be provided by waste heat from the roaster and the acid plant. The sulfate crystals could then be shipped to a refinery, such as Port Nickel, for Ni-Co recovery. The above approach eliminates the need for manganese removal, Zn/Cu precipitation, and Zn-Ni-Co scavenging.

Anschutz piloted its milling circuit using 225 tons of underground ore, and in a separate test, 100 tons of tailings. Feed rate to the pilot plan was 5 tph. Roasting conditions for the material sent to Sherritt Gordon for refining were adjusted to maximize cobalt extraction during the subsequent pressure leach. Cobalt extraction during a commercial operation should average about 95%. Copper, which is readily sulfated under these same conditions, should show extractions consistently exceeding 95%. Nickel has a greater tendency to form ferrite, $(Ni,Fe)_2O_4$, and refractory nickel oxide. Therefore, nickel extraction will be lower, typically 85%. The large "unaccounted" category in Figures 6-9, -10, and -11) is primarily due to removal of samples for thickening tests and analysis. "Unaccounted" should actually be about 2%. Overall recovery of Co, Ni, and Cu from ore to product (Figures 6-9, -10, and -11) is projected at 75%, 70% and 90%, respectively.

If one were forced to choose between investing in Madison or Blackbird cobalt, with the decision based only on process technology, Madison should be given preference. This preference is based on Madison's production of a selenium-free product. A secondary consideration is the low arsenic content of Madison ore, as compared to Blackbird. The complexity of the two processes is equivalent. Both could be simplified in order to generate impure Co-Ni intermediates for shipment to a central refinery. Such a refinery could well be Port Nickel. To make Port Nickel a versatile central processing plant, a modernization program would be required using available technology to upgrade the plant for treating feeds having a wide range of Co, Ni, Cu, Se, S, Fe, Pb, As, and Zn content.

Cobalt Recovery from Missouri Lead Operations

Most of the published papers on cobalt recovery from Missouri Lead operations are studies completed by the USBM. Missouri Lead ore typically assays 6% Pb, 1% Zn, 0.3% Cu, 0.02% Ni, and 0.015% Co. Total probable ore reserves are 325 million tons (Paulson, 1980). During the beneficiation and smelting of the ores, cobalt and nickel are lost in the flotation tailings and slag (Figure 6-12).

The most direct route to recovery of a portion of the cobalt (and nickel), as developed by the USBM Rolla Research Center, has been summarized by Peterson et al. (1981). The method results in recovery of siegenite from chalcopyrite concentrate that assay 28% Cu and 0.5 to 1.5% each of nickel and cobalt (Clifford, 1978). The process consists of closed-circuit (hydrocyclone) grinding of the chalcopyrite concentrate, which is produced as a byproduct from flotation upgrading of galena ore. Flotation reagents diethyl dithiophosphate (collector), sodium cyanide (depressant), and methyl isobutyl carbinol (frother) are added to the hydrocyclone discharge. An enriched copper concentrate is subsequently recovered during cleaner flotation (Phillips, 1979). A Ni-Co concentrate (5.0% Ni, 3.7% Co, 17% u, 12% Pb, and 1% Zn) is recovered in the laboratory as the sink product from scavenger

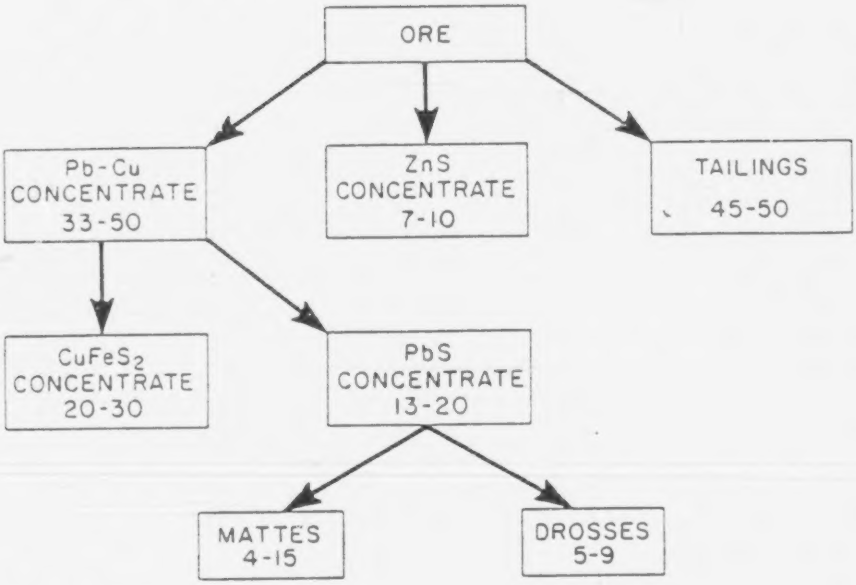

ORE
Pb-Cu CONCENTRATE 33-50
ZnS CONCENTRATE 7-10
TAILINGS 45-50
$CuFeS_2$ CONCENTRATE 20-30
PbS CONCENTRATE 13-20
MATTES 4-15
DROSSES 5-9

FIGURE 6-12 Distributions of Siegenite When Processing Missouri Lead Ore

flotation. Cobalt and nickel recovery is 80% to 85% from the copper concentrate, which contains about one-fifth of the quantity of these values in the lead ore. Pilot-plant testing of this Ni-Co recovery technology was carried out at Cominco American's Magmont mill (Sharp, 1976) using a 130-lb/hr bleed stream from the copper circuit. A 5% Ni, 3% Co concentrate was produced with 80% recovery (Clifford, 1978). Further piloting is presently underway at the Magmont mill, which is reportedly attaining higher grade Ni-Co (10% of each) and lower levels of lead (6% to 8%). The level of Ni-Co recovery had not yet been determined as of this writing.

The USBM publications focus on processing existing lead-plant outputs for Ni-Co recovery. These outputs not only include byproduct chalcopyrite concentrate, but also blast-furnace slag, lead-smelter mattes and drosses, and lead-mill tailings (Paulson et al., 1980). However, there is also opportunity within these plants to modify the distribution of cobalt and nickel in a manner that facilitates subsequent recovery. For example, a somewhat greater quantity of iron could be allowed to float with the lead, which reportedly substantially increases (perhaps doubles) the quantity of cobalt that reports to the chalcopyrite concentrate. the higher subsequent overall cobalt-nickel recovery must be balanced against increased blast-furnace operating cost due to the higher iron content of the furnace charge. Implementing such modifications of existing Missouri lead mill operations has, as its incentive, the recovery of about one million annual pounds of cobalt.

Nickel-cobalt concentrates collected as byproduct from Missouri lead mills contain substantial quantities of lead (typically 2% to 10%). Technology for handling such material has been published by Hoboken-Overpelt (British patent, 1979). This technology, or modifications of it, could almost certainly be incorporated into a domestic processing facility (e.g., Port Nickel). One such modification would be installing a top-blown rotary converter at a central location in the Missouri Lead Belt for smelting the cobalt-rich concentrates.

There is also potential for floating a pyrite concentrate from Missouri lead-mill tailing output, containing most of the nonferrous metal values in the tailings. Commercially proven processes are available to roast this pyrite to iron oxide, followed by chlorination of the oxide to remove and concentrate the nonferrous metals. The iron oxide is then pelletized and used as iron blast furnace feed. The nonferrous chlorides are refined into Cu, Ni, Co, Pb, Zn ..., as well as precious-metal products. Economics depend not only on metal sales, but also on having a market for the sulfuric acid produced as a byproduct of roasting the pyrite.

A problem with plant research is that it tends to focus on short-term goals, and to skirt projects that require non-essential capital investment. A problem with the USBM research program is its limited access to plant mass balances, particularly in applications that involve changing these balances. Perhaps these shortcomings could be minimized by the USBM's sponsoring a joint industry-bureau project directed toward byproduct Co-Ni recovery in Missouri, similar to the Reno Research Center's cooperative venture on hydrometallurgical processing of galena concentrates. Ideally, the project would interface with the Missouri Lead industry as a whole. Present work deals with the plants individually, which limits the ability of the rest of the industry to provide expertise to a given project. One could envision such an effort leading to pilot plant testing of alternative flowsheets over a three-year period, followed by commercial implementation of the most straightforward approaches some two years later.

MANGANESE

As discussed in Chapter 5, the domestic onshore manganese deposits best suited for development on a significant scale are those of Minnesota's Cuyuna Range and those of Aroostook County in Maine, but prospects for commercial production of manganese from these deposits are poor. Conclusions from the National Materials Advisory Board (NMAB) comprehensive 1976 report, "Manganese Recovery Technology," include the following:

1. The United States has no domestic resources from which the manganese products suitable for steelmaking can be recovered economically, and no facilities exist, nor are planned, for recovering manganese from the low-grade manganese deposits that do exist.
2. The U.S. land-based manganese resources of significant size are very low in grade, and should not be developed except in a dire emergency. While extensive development work has been done on two manganese extraction processes, questions concerning performance, cost, and environmental impact remain to be resolved before full-scale operation could be assured.
3. Use of physical beneficiation methods (e.g., flotation, gravity concentration, or magnetic separation) to produce concentrates suitable for direct production of ferromanganese from any of the major domestic deposits does not appear feasible at this time.

The above conclusions appear to be as valid in 1982 as they were in 1976. An information circular (I.C. 8889) issued by the USBM in 1982, "Manganese Availability, Domestic," further concluded:

> Domestic deposits are submarginally subeconomic, requiring greater than 1.5 times the current price of manganese, or a major cost-reducing advance in technology, in order to be considered for development. The possibility of a major technological breakthrough appears unlikely at the present time, and the eight majaor domestic manganese deposits would probably not be developed except in an extreme national emergency.

Conclusions and Recommendations

1. Stockpiles of high-grade manganese ore and ferromanganese adequate to cover possible upsets in supply should be maintained. Lead time for

significant production of ferromanganese from domestic sources is at least three and one-half years. Priority should be given to importing manganese nodules from Mexico (i.e., Minera Autlan), thus supporting this nearby manganese-resource base. These imports should be smelted in domestic furnaces to ferromanganese for stockpiling, providing government subsidy if necessary.

2. If stockpiling of manganese materials does not satisfy this domestic strategic-metal need, the preparation work should be completed to allow the Cuyuna deposits to be brought on line three and one-half years after notice of a manganese supply emergency is served.

3. At least 1.5 times the current price of manganese will be required before production of substantial tonnages of ferromanganese from domestic deposits becomes economic using existing technology. The best known deposits are in Minnesota's Cuyuna Range. The ammonium carbamate process appears to be the most economic of the known processes. A key to economic utilization of disseminated low-grade deposits is beneficiation technology. There appear to be substantial savings attainable by improving comminution and fine-particle concentration processes. These areas are prime targets for federal R&D funding.

4. Domestic manganese sources are already being used to supplement feedstock to domestic iron blast furnaces. The amount of Cuyuna manganese concentrate that could be blended with domestically consumed iron ores should be estimated a key constraint for such blending will be the manganese-to-impurity ratios for a given grade of manganese concentrate, especially with respect to titanium, magnesium, and silica (St. Pierre, 1982). Blending not only avoids subsequent hydrometallurgical processing, but also eliminates the need for electric-furnacing to ferromanganese.

Historical Domestic Production of Manganese

During the latter part of the 19th century, the United States produced sufficient manganese from domestic deposits to meet most domestic needs. Since 1900, manganese consumed by the growing domestic steel industry has vastly out-stripped domestic production. Despite a variety of government incentive programs, domestic production was only 23% of consumption during World War I, 13% during World War II, and 8% during the Korean war (NMAB, 1976). In 1980, there was neither production nor shipment of manganese ore, which by definition contains 35% or more manganese, in the United States.

Small quantities of low-grade ferruginous manganese ores of concentrates containing up to 35% manganese were produced during 1980 in New Mexico and in the Cuyuna Range of Minnesota (DeHuff et al., 1980). They were used as low-grade feeds to iron blast furnaces (Kilgore, 9182). This manganese supplements the small but significant amount of manganese contained in many iron ores. The manganese impurity in iron ore reportedly provides about 225,000 annual tons of manganese to the United States, i.e., about 20% of our country's annual manganese consumption (St. Pierre, 1982). Only about 25,000 tons of manganese contained in ferruginous ores (averaging about 14% Mn) and manganiferous ores (averaging about 8% Mn) were mined domestically in 1979-1980.

Physical Beneficiation of Manganese Ores

The domestic deposits of greatest potential for manganese production occur in Minnesota's Cuyuna Range. They are sedimentary in origin, and are in an area in which iron ores with various manganese contents have been mined for decades. Mining experience is thus good. There is also a sound infrastructure of power supply and transportation (NMAB, 1976). Furthermore, the USBM has piloted two hydrometallurgical manganese recovery flowsheets (SO2-roast and ammonium carbamate) using Cuyuna feedstock. Estimated reserves are 20 million tons of contained manganese at 8% grade, which should be sufficient to satisfy projected U.S. demand for at least a decade. Of the 20 million tons, 12 million are contained in nine individual bodies with average grades ranging from 6% to 10% Mn.

A typical flowsheet for beneficiation of sedimentary manganese ore may include crushing, washing, screening, jigging, and tabling. High-intensity magnetic separation is sometimes used for concentrating the fine fractions. Cuyuna Range manganese mineralization consists of manganese carbonates, manganiferous limonite (an iron-manganese oxide), manganite ($Mn_2O_3\ H_2O$), and psilomelane ($(Ba,H_2O)_2Mn_5O_{10}$). Gangue minerals are predominantly quartz (SiO2), iron silicates, iron carbonates, magnetite (Fe_3O_4), limonite, and some pyrite (FeS_2).

Physical beneficiation, if applicable, would be the simplest and least expensive means of recovering manganese from domestic sources (NMAB, 1976). A breakthrough in beneficiation technology would be the single most significant factor for improving the economic status of known domestic manganese deposits (Kilgore et al., 1982). A key problem at present is liberation of the manganese mineralization. At Cuyuna (and Aroostook) the minerals are so fine-grained and interlocked that the manganese minerals are not liberated adequately when the ore is ground to a reasonable particle size. Therefore, high-grade products have not been obtained with good recovery of manganese.

The 1976 NMAB report points out that prior work has been directed toward producing ore-grade (greater than 35% Mn) concentrates, resulting in unacceptable manganese recovery. The authors proposed a modest research program with the objective of upgrading 8% Mn ore to a 25% Mn concentrate at good recovery. The savings in the cost of the subsequent hydrometallurgical operation might exceed the cost of concentration plus the cost of mining additional ore to make up for manganese losses to the concentrator tailings. This proposal has merit, and is supported by prior art, as described below.

Work by Bogdanov (1975) on low-grade sedimentary manganese ore from Russian deposits progressed along the above lines. Low-grade manganese carbonate (11% Mn) was reportedly upgraded to 26% Mn using flotation, followed by high-intensity magnetic separation. Use of "froth separation," in which the feed enters the froth from above instead of from below, was claimed to be particularly efficient. Reportedly, "considerable economic benefits" accrued after conventional flotation was replaced by froth separation at two Russian plants. Feedstock included slimes containing 30% to 50% minus 20-micron material. Plant operation, according to Bogdnov, utilized high-intensity magnetic separation for upgrading the froth-separation concentrates.

Incentive for developing technology to liberate fine interlocked minerals has been addressed recently by J. M. Williams (Assistant Director, Los Alamos National Laboratory). He points out that the energy used for comminution (crushing and grinding) of approximately one billion tons of U.S. ore during 1981 was approximately 1% efficient in increasing the surface area of the ore. He refers to a 1981 report by the Committee on Comminution and Energy of the NMAB that includes recommendations for innovations in comminution device design, classifier device design, process control, grinding additions, and materials of construction.

Research on beneficiating low-grade manganese ores continues in Russia, as evidenced by their recent description of possible advantages of incorporating vibration into a flotation operation (Tomov, 1982). Perhaps more significant is that several domestic R&D programs are now focusing on the problems of handling and concentrating fine particulates, e.g., minus 10-micron slimes. Recent commercial applications for concentration of values from slimes, such as selective flocculation and fine-bubble flotation, perhaps have application for domestic manganese-ore beneficiation.

Processing of Cuyuna Manganese Concentrates

Because 90% of the manganese consumed in the U.S. is used in the manufacture of iron and steel products, a portion of the product from processing Cuyuna ore should be compatible with this major end use. The 1976 NMAB report suggests that minimum quality for high-carbon ferromanganese be at least 70% Mn and less than 0.5% total Cu-Ni-Co-Mo. The recommended process for Cuyuna, ammonium carbamate leaching, produces agglomerated manganese oxide superior in chemical composition to world-class high-grade ores. In fact, oxide produced from Cuyuna feedstock in a "semi-work plant" (i.e., a large pilot plant) was sufficiently pure to be used for commercial preparation of battery-grade MnO_2 (Welsh et al., 1975). Note that even if sufficient domestic agglomerated manganese dioxide were available, there could be a shortfall of ferroalloy production capacity (NMAB, 1976).

The ammonium carbonate process dissolves manganese preferentially from reduced calcine to iron. The calcining operation (450°C) utilizes carbon monoxide gas to reduce manganese compounds to carbamate-soluble MnO and carbamate-insoluble Fe_3O_4. Subsequent leaching of the calcine utilizes an ambient-temperature solution that is 18 moles in ammonia and 3 moles in carbon dioxide. Manganese dissolves as an unstable carbonate complex,

$Mn(NH_3)ONH_4O_2CNH_2$, which is decomposed to insoluble manganese carbonate by passing steam through the pregnant liquor. Steaming drives off excess aqueous ammonia and O_2, which is recycled to leach more calcine. The manganese carbonate is roasted at 1100°C to produce Mn_3O_4 product (Norman and Kirby, 1963). This oxide (sinter) typically assays 65% Mn, 3% SiO_2, 1% Fe, 0.02% Ni, 0.14% Co, and less than 0.02% Cu.

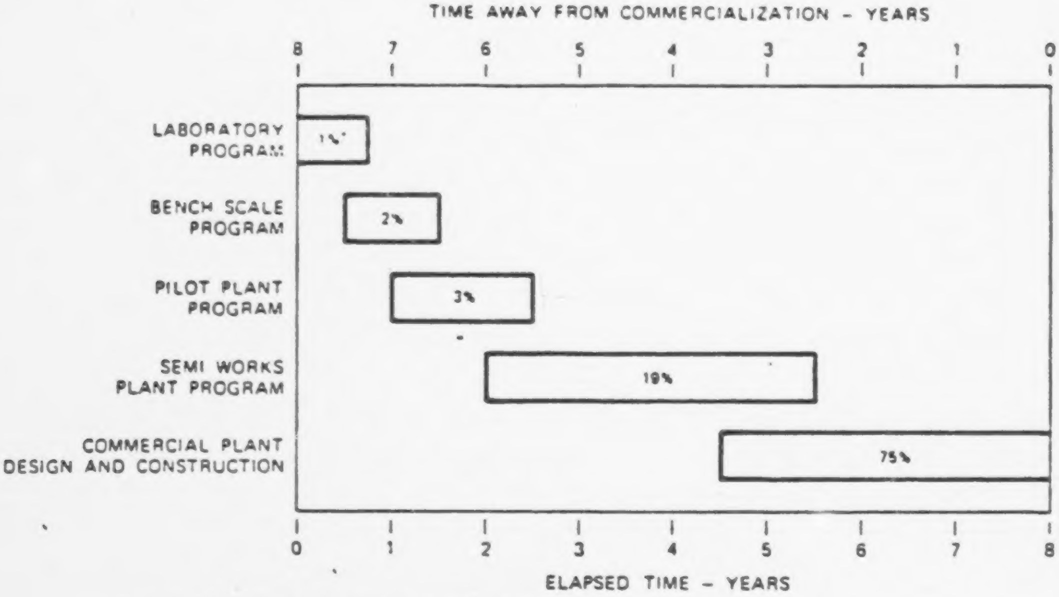
The 1976 NMAB report also recommends, as an alternative to the carbamate process, the sulfur dioxide roast process. This latter process uses twice as much energy as the ammonium carbonate process. Due to the escalation of energy costs since 1976, the carbamate route was therefore given preference (Kilgore and Thomas, 1982). Alternative non-hydrometallurgical-based processes are smelting and vapometallurgy. Smelting is not advisable due to the low grade of domestic ores. Furnace utilization is poor, and flux consumption, energy consumption, and manganese loss to slags are all high. Vapometallurgy approaches include high-intensity arc decomposition of ore into MnO and SiO^2 for subsequent leaching, chlorine volatilization of $MnCl_2$ and $FeCl_2$ from the ore followed by hydrometallurgy, and smelting the ore to low-grade ferromanganese followed by oxygen blowing to volatilize manganese. The vapometallurgy processes studied to date have not proceeded significantly beyond bench-scale experiments. These latter processes do not appear sufficiently technically or economically attractive to provide incentive for further research or development (NMAB, 1976).

Lead Time for Domestic Manganese Production

The 1976 NMAB report details time (and cost) estimates to achieve manganese oxide sinter production from the Cuyuna deposits. Six to seven years would probably be required to bring the present state-of-the-art in mining, beneficiation, and carbonate processing to a commercially viable operation. If pilot plant, semi-works plant, and design work were completed prior to the "emergency," a commercial plant could possibly be on-line within three and one-half years. Refer to Figures 6-13 and -14, as reproduced from the 1976 NMAB report. Factors that could extend this minimum time estimate include less-than-rapid delivery of mining equipment, inadequate availability of materials and production capacity for process equipment, fabrication, and legal or environmental delays. The three and one-half year minimum time estimate also requires prior completion of (1) mine plans, (2) detailed process-plant engineering, (3) engineering plans for infrastructure, (4) environmental impact statements, (5) contracts with property and process-patent owners, and (6) all legislation needed to permit immediate action upon authorization of the concerned federal agency or officer.

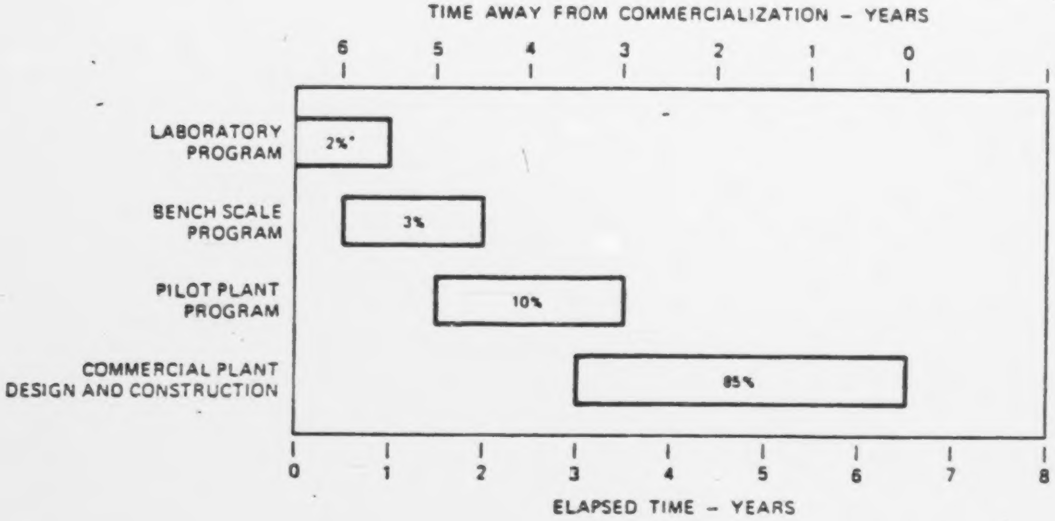
Solution Mining of Manganese Ores

In the 2nd SME-SPE International Solution Mining Symposium (1981), R. B. Bhappu (Mountain States Mineral Enterprises, Inc., Tucson) described his belief in the importance of metal recovery via solution mining. One of the metals discussed for recovery by solution in-situ mining techniques was manganese. The leach liquor recommended by Bhappu was aqueous sulfur dioxide. Solar evaporation was proposed to concentrate the pregnant liquor sufficiently to crystallize manganese sulfate, which would be purified, then calcined to produce oxide sinter. Process details and economics were not given. Leaching chemistry and pregnant liquor treatment would presumably be modeled after that described in the Henn et al. (1968) review of SO_2 processes for recovering manganese from domestic resources.



*% TOTAL COST TO DEVELOP AND COMMERCIALIZE NEW PROCESS. (Escalation not included)

FIGURE 6-13 Time and Cost to Commercialize New Process When Semi-Works Plant Is Required.



*% TOTAL COST TO DEVELOP AND COMMERCIALIZE NEW PROCESS. (Escalation not included)

FIGURE 6-14 Time and Cost to Commercialize New Process When Semi-Works Plant Is Not Required.

In-situ solution mining has been developed for recovering a number of substances including salt, potash, sulfur, copper, and uranium. The most successful example that involves metal recovery is the Miami copper operation at Miami, Arizona (Fletcher, 1971). For the past 20 years, copper has been recovered by in-situ leaching of block-caved ore. Leach liquor is collected from underneath the ore zone using passages left from old underground workings. The writer knows of no commercial in-situ metal mine presently operating (December 1982) in the absence of an old underground or open-pit mine. The uranium in situ leaching operations operated during the past decade used "virgin" sites, but were shut down after the price of U_3O_8 dropped from about $40/lb to less than $20/lb. Further details on this subject are reportedly contained in the write-up for recently completed USBM Contract No. J0295032.

Important for in-situ mining of metals is requisite controlled flow of leachant through large volumes of ore. The key words are requisite controlled flow. For example, a number of domestic corporations have commercially recovered U_3O_8 via in-situ mining from domestic deposits. Although significant problems were encountered in obtaining and maintaining ore-body permeability, the most difficult technical barrier was attaining complete recovery of the leach liquor. Unrecovered leachant not only lowers metal recovery, but also endangers local groundwater and hinders restoration of the well field. A number of the domestic U_3O_8 well fields commercially exploited during the uranium boom of the 70's have yet to be restored. Details on the technical and economic problems associated with restoration may be available through In-Situ, Inc., a consulting firm run by C. R. McKee in Laramie, Wyoming. Mr. McKee should also be able to comment on the probability of achieving an economically viable manganese in-situ mining operation. This writer recognizes such a goal as an opportunity but has no knowledge of the associated economics. For a general description of the economics of in-situ mining of copper, gold, and uranium, refer to the 1976 Lewis paper. A detailed bibliography on in-situ leaching was prepared by W. C. Larson at the USBM Research Center in Minneapolis in 1976-76.

Biochemical Leaching and Molecular Biology

The role of sulfate-reducing bacteria in the formation of sulfide ores was suggested early in this century. Utilization of bacteria to accelerate commercial extraction of metals from waste rock probably began in the late 1940's. The primary commercial application at present is the leaching of copper from overburden removed during open-pit mining. Bacterial leaching of manganese ores was reported by E. C. Perkins in 1958. The question arises in the early 1980's: Can the molecular biologist generate a super-bug to recover manganese from low-grade domestic deposits?

Feedstock Considerations

The medium in which the leaching is to occur must have available the nutrients and other substances necessary to sustain growth of the microorganisms. These requirements include water, carbon, either the presence or absence of oxygen, nitrogen, phosphorus, and certain trace elements (Brierley, 1978).

Reactor Constraints

In commercial reaction systems, competition from unwanted bacteria must be avoided. Therefore, the ideal microorganism for an ore body will probably be a naturally occurring life form rather than one that is synthesized. The requirement for sustained microbial activity by a given organism restricts to a narrow range the operating temperature, pH, dissolved oxygen concentration, substrate, and the concentration of reaction products (Tong, 1979). Microbes, because of their small size, have a large surface/volume ratio and hence are rapidly affected by changes in their environment. These environmental constraints, which must be met to avoid mutation or death of the organisms, have traditionally been achieved in batch reactors, some as large as 500,000 gallons (Burns, 1981).

One argument for the use of biological reactors is that they can be performed at or near room temperature. In fact, most microorganisms require near-ambient conditions to survive. The cells and enzymes that carry out the reactions require temperature control that is more exacting than has been conventional in the chemical industry. Futhermore, oxidation reactions generate heat. A major problem is economic removal of this heat from high-volume systems at temperatures near 25^{o}C (Brooks, 1981).

Removal of Reaction Products

The reaction products from enzyme-catalyzed reactions often inhibit enzyme reactivity. For instance, certain metallic products of leaching are toxic to T. ferrooxidans, an important bacteria for oxidizing ferrow iron to ferric iron , which in turn oxidizes base metal sulfides. Examples include traces of silver, molybdenum, mercury, and cadmium. Even the base-metal products of biological leaching processes, such as copper, become toxic to microorganisms above a certain base-metal concentration. This problem is minimized by giving naturally-occurring bacteria time to become acclimatized to increasingly large concentrations of the metal-ion product (Bruynesteyn, 1971).

Conclusion

Biotechnology provides a potential means for improving the economics of manganese recovery from domestic resources. This potential extends beyond leaching, and includes manganese absorption and concentration, precipitation, mine and refinery waste reclamation, and even production of reagents. The microorganisms have rather specific environmental constraints that are seldom met at the ore body of choice. Prime candidates for biochemical processing of manganese would be naturally occurring bacteria. Constraints of feedstock, energy source, development costs, scale-up and process control are pressing problems in economic use of microorganisms synthesized via recombinant DNA. Traditional genetic engineering--development of more efficient "bugs"--utilizes classical procedures of selection and mutation. This latter route appears to the writer to hold greater promise in this century for extractive metallurgy than does "gene splicing."

Manganese Production in Mexico

If one confines his study to domestic on-shore manganese deposits only, the picture is indeed bleak. The situation improves if one changes the focus on "on-shore manganese deposits." Now one can look south to Mexico's Compania Minera Autlan operation, producing 500,000 annual tons of manganese oxide nodules (40% Mn) from ore containing 27% manganese. Two 33,000-kva electric furnaces are used to convert a portion of these nodules to ferromanganese (100,000 annual tons). Minera Autlan ore reserves are reportedly 25-MM tons proven, 50-MM tons probable, and 1,500-MM possible. Minera Autlan reportedly needs financial support for possible expansion, support that presumably could result in the funneling of a significant portion of its ferromanganese production into the U.S. stockpile.

Minera Autlan's plant is located at Tampico, which is about 300 miles south of the U.S. border. The plant's principal access to U.S. markets is by ship up the coast to Mobile, where the company has additional electric-furnace capacity. Alternate transport is by rail. Half of the ferromanganese produced at Tampico contains 72% to 74% Mn, is consumed in Mexico, and is produced without nodule enrichment. Export-grade ferromanganese, 78% to 80% Mn, requires the addition of imported high-grade ore. In a national emergency, the 72% to 74% Mn ferromanganese should be a suitable substitute for the higher grade material.

Mr. Ed Mancke, a consultant who works with Minera Autlan, is knowledgeable on manganese metallurgy, and has recently completed a detailed review of manganese process metallurgy for Minera Autlan. A copy of this review could be purchased, states Edgar B. Mancke, P.E., telephone (717) 897-6278, Portland, PA.

Work at Arthur D. Little, Inc.

Arthur D. Little, Inc. (Cambridge, MA), under project leadership by Dr. S.V. Margolin, will complete an $86,000 USBM contract during 1983 on "Recovery of Manganese from Low-Grade Domestic Sources." The purpose of this study is to identify the technical and economic viability of processes proposed and/or developed since 1950 for recovery of manganese products from domestic low-grade deposits. Dr. Margolin has informed me that his primary thrust will be to show that developments in process technology during the past 20 years improve the economics of manganese recovery from the Cuyuna and Aroostook deposits. These breakthroughs include improvements in equipment and process control, for example, as related to fluid bed roasting and decomposition of process intermediates (e.g., metal nitrates and chlorides).

One could assume that developments in solvent extraction, ion exchange, and membrane technology will be part of the study. Also included will be any spin-off from innovations developed for processing manganese nodules.

Almost certainly Dr. Margolin will show savings in overall processing costs when applying the test of today's technology to one or more of the 15 processes selected for study. It is unfortunate that his contract does not include the study of applicable advances in beneficiation, such as selective flocculation, special agglomeration, agglomerate flotation, fine-bubble flotation, and carrier flotation. Arthur D. Little is well qualified to do such an analysis. Furthermore, the sponsor of the project (i.e., the USBM) recently concluded in their sensitivity analysis of costs for domestic manganese production that beneficiation headed the list of opportunities for decreasing these costs.

NICKEL

Introduction

The two most important domestic primary sources of nickel are Minnesota's Duluth Complex and Oregon-Washington-California laterites. Byproduct nickel in the form of nickel sulfate is produced during the electrorefining of copper. Significant quantities of nickel may also become available as an important byproduct from cobalt recovery in the Missouri Lead Belt, as discussed in the section entitled "Cobalt."

The most attractive domestic nickel deposit is the Duluth Complex both from the standpoint of nickel tonnage and economics. The nickel occurs as sulfide associated with larger quantities of copper and iron sulfides. Precious metals and cobalt are important byproducts. The host rock, gabbro, is a granular basic igneous rock containing important quantities of sodium aluminum silicate that could be a useful source of alumina. The Cu-Ni sulfides could be separated from the comminuted ore by froth flotation, producing a copper and a copper-nickel concentrate, which are smelted and refined to produce primary-metal products. This subsection deals with recovery of copper, nickel, and cobalt. The PGM (platinum group metals) process technology is discussed in "Platinum."

The Oregon-Washington-California laterites contain nickel as oxide, which requires a very different metallurgical approach for its recovery. A key difference is that little can be done via physical beneficiation to concentrate oxide nickel. Smelting or leaching operations must therefore handle most of the gangue (waste) portions of the ore. This added tonnage usually results in higher production costs than occur with sulfide ores. Nickel can be recovered from oxide ores by direct smelting, reduction roasting followed by leaching, or leaching alone. Leaching has the advantage of maximizing recovery of byproduct cobalt values. Domestic nickeliferous laterite does not contain precious metals.

Conclusions

1. The greatest potential for high-tonnage production of nickel (e.g., 20,000 tpy Ni) is in Minnesota's Duluth Complex. Before it becomes attractive for development, the world's present nickel-production capacity must first come back on line. Second, nickel and copper prices should be at least $1.50/lb Cu and $4/lb Ni, in 1983 dollars.

2. The Port Nickel refinery at New Orleans could be modernized to allow production of highly refined nickel and cobalt (e.g., cathodes). Feed sources

could include concentrates from Minnesota, Blackbird (Idaho), and the Missouri Lead Belt, and open-market purchases of miscellaneous Ni-Co feedstocks.

3. Cobalt byproduct production from the Duluth Complex, based on 100,000 tpy copper production, could be as high as 800 tpy. This recovery depends on efficient cleaning of nickel and copper converter slags. The largest single cobalt loss (1800 tpy) is to the flotation tailings.

4. Hanna's nickel operation in Riddle may remain shut down due to increasing power costs and lower ore grade. In the event of a prolonged plant shut down, consideration should be given to its "mothballing" so that in an emergency it could serve as a processing facility for domestic lateritic nickel ores.

5. The California nickel-cobalt project, as proposed for Gasquet Mountain has not demonstrated that it measures up to the tonnage, ore grade, or technology requirements recognized as necessary for economic recovery of metal values from lateritic nickel.

Minnesota's Duluth Complex

The Duluth Complex is recognized as the largest known nickel resource in the nation. Copper is a very important co-product; byproduct precious metal values are substantial. There is also potential for recovering significant cobalt and anorthite (calcium aluminum silicate) byproducts (Lawver, 1975). A 10,000-ton ore test sample mined by INCO at its Spruce Pit site near Ely was successfully upgraded into a bulk Cu-Ni concentrate at INCO's Creighton mill. Detailed test work and financial analyses, as published by engineers at the University of Minnesota's Mineral Resources Research Center, also show that the Duluth Complex deposits have great long-term promise.

Copper values in the Duluth Complex are typically 0.43%, and are present as chalcopyrite ($CuFeS_2$) and cubanite ($CuFe_2S_3$). The cubanite is sufficiently abundant that copper concentrates are limited to about 25% copper. Nickel in the ore is typically 0.15%, and is present as pentlandite ($(Fe,Ni)_9S_8$), along with much larger quantities of pyrrhotite ($Fe_{1-x}S$). Most of the pyrrhotite is not magnetic. Some nickel is also tied up with the gangue minerals, particularly olivine (magnesium silicate), which contains about 0.08% Ni. This tie-up limits nickel recovery into a sulfide concentrate to less than 75% and typically to about 60% to 70%. Approximately one-quarter of the nickel and perhaps even more of the cobalt in the deposit occurs in olivine. Metal values associated with olivine report to tailings. Bacterial leaching may be of some assistance. The state of Minnesota, INCO, and AMAX have given considerable thought to mine planning for Ni-Cu production from Duluth Complex.

A key decision in selecting a beneficiation flowsheet for Duluth Complex ore is whether to use bulk flotation or differential flotation. INCO, Kennecott, AMAX, and the state of Minnesota have completed test work on both approaches. The most detailed published data are those of the state of Minnesota, particularly Lawver (1975) and Reid (1979). The Lawver study is based on bulk flotation, a decision that was made to simplify the associated economic evaluation of the resource. In actual practice, the bulk flotation concentrate would almost certainly be beneficiated further to produce separate copper and copper-nickel concentrates, as shown in Figures 6-15 and -16. The differential flotation approach allows direct delivery of Cu and Ni-Cu concentrates to separate copper and nickel smelting furnaces, which preferably would be constructed side-by-side in northern Minnesota.

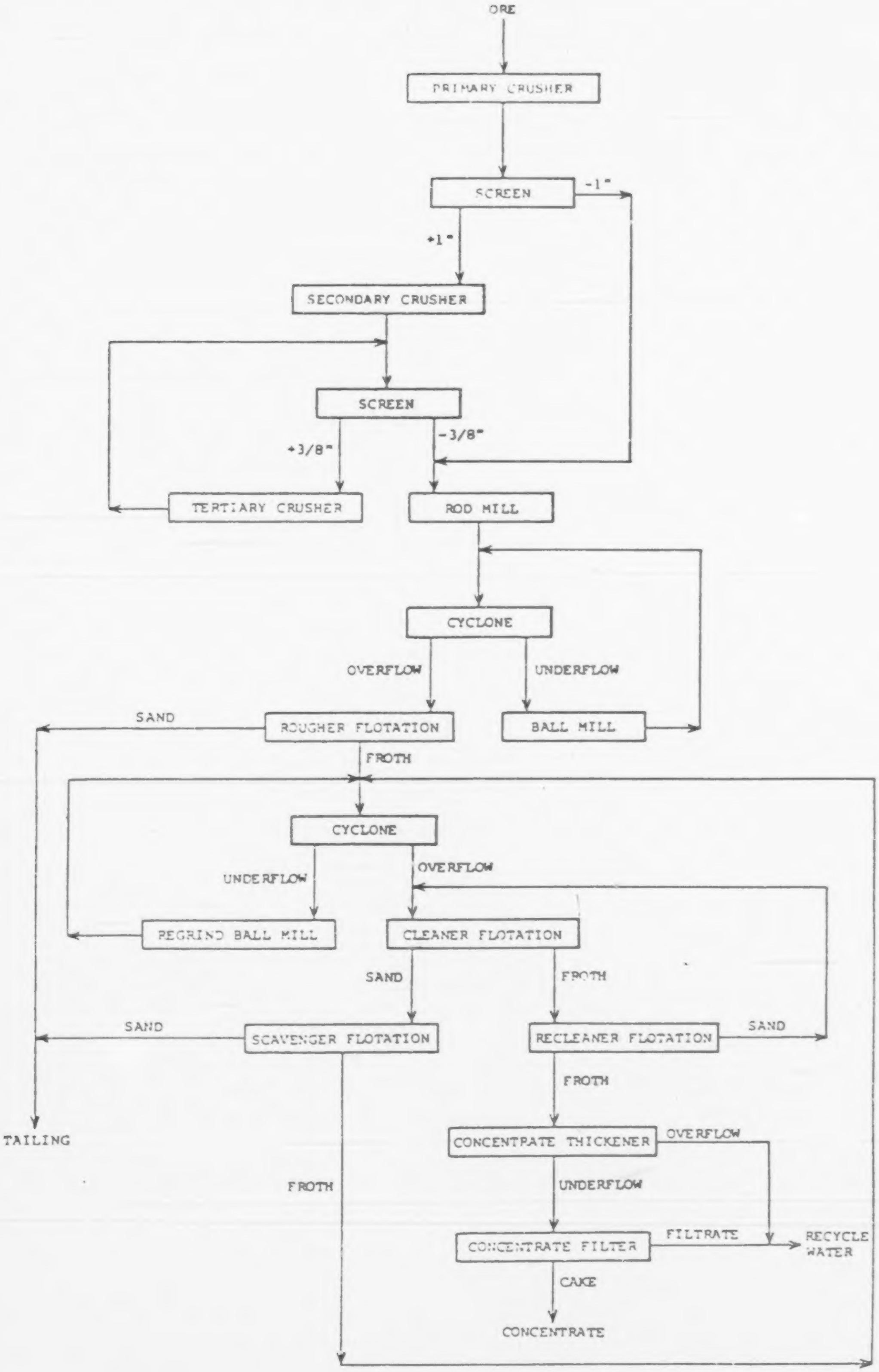

ORE
PRIMARY CRUSHER
SCREEN
-1"
+1"
SECONDARY CRUSHER
SCREEN
+3/8"
-3/8"
TERTIARY CRUSHER
ROD MILL
CYCLONE
OVERFLOW
UNDERFLOW
SAND
ROUGHER FLOTATION
BALL MILL
FROTH
CYCLONE
UNDERFLOW
OVERFLOW
REGRIND BALL MILL
CLEANER FLOTATION
SAND
FROTH
SAND
SCAVENGER FLOTATION
RECLEANER FLOTATION
SAND
FROTH
TAILING
CONCENTRATE THICKENER
OVERFLOW
FROTH
UNDERFLOW
CONCENTRATE FILTER
FILTRATE
RECYCLE WATER
CAKE
CONCENTRATE

FIGURE 6-15 Bulk Flotation of Minnesota Duluth Gabbro Ore (Lewis, 1978).

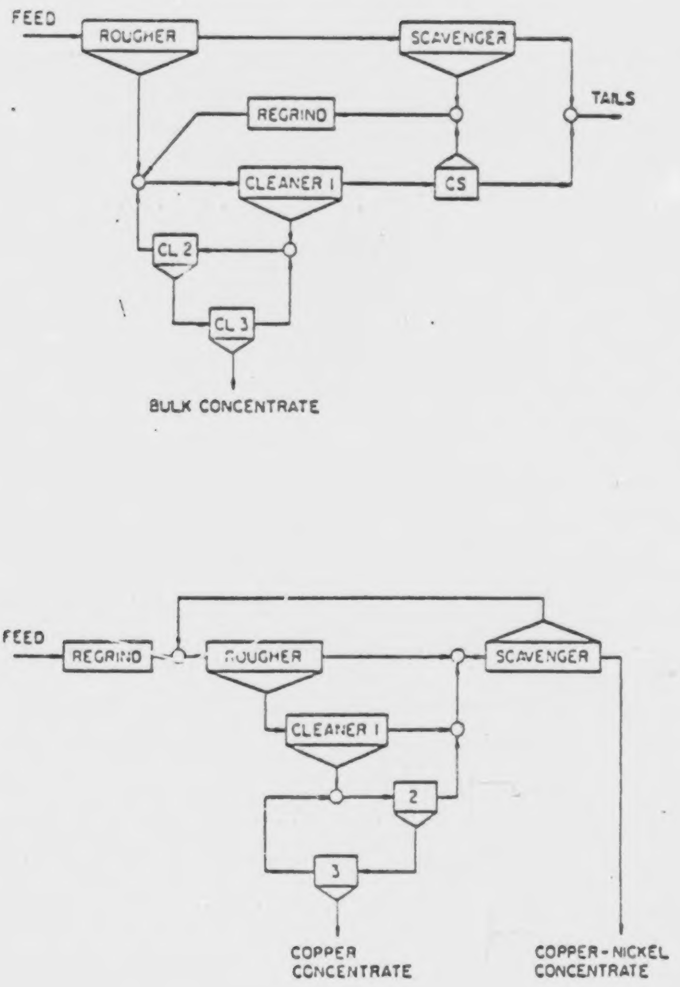

FEED
ROUGHER
SCAVENGER
REGRIND
TAILS
CLEANER 1
CS
CL 2
CL 3
BULK CONCENTRATE
FEED
REGRIND
ROUGHER
SCAVENGER
CLEANER 1
2
3
COPPER CONCENTRATE
COPPER-NICKEL CONCENTRATE

FIGURE 6-16 Differential Flotation of Duluth Gabbro Bulk Concentrate (Reid, 1979). The feed to the differential flotation circuit (bottom) is produced by the bulk flotation circuit (upper).

Copper recovery by flotation into the copper concentrate should be about 90%, which is normal for a low-grade sulfide ore. Grade of the copper concentrate should be about 23% Cu and about 0.7% Ni. This nickel represents about 15% of the nickel recovered from the ore. Nickel recovery from the ore by flotation should be about 65%. The resulting nickel-copper flotation concentrate should assay about 4% Cu and 4.3% Ni. The copper content of this Ni-Cu concentrate represents about one-third of the copper recovered from the ore.

Cobalt recovery during the flotation of copper and nickel values is low, on the order of 40%, based on work performed by the state of Minnesota, the USBM, and AMAX. One factor contributing to the loss is the presence of cobalt in the non-floating olivine. The ratio of Co/Ni in the olivine is apparently significantly higher than the Co/Ni ratio in the bulk ore. Th Ni-Cu concentrate should assay about 0.10% Co, as concentrated from ore assaying 0.012% Co. This cobalt represents 85% of that recovered from the ore. The remaining recovered cobalt will report to the copper concentrate.

A description of production of bulk Cu-Ni concentrate, as would be produced after open-pit mining of Duluth Complex, is given below (Lewis, 1978).

> Ore would be delivered to a primary gyratory crusher near the concentrator. Crusher discharge would be about minus 8-inches, and would be conveyed to secondary and tertiary cone crushers. The tertiary crushers would be closed-circuited, with screens, to hold the primary product to minus 3/8-inch for feed to conventional rod-and-ball mill grinding. Screened discharge from primary and tertiary crushing would go into conventional rod mills using open circuit grinding to minus 35-mesh. The overflow would then pass to a closed-circuit ball mill, grinding to 80% minus 200-mesh. Oversize product from the ball mills would then be sent to regrind mills to an 80% minus 325-mesh product. FLotation-plant layout would incorporate 500-cubic-foot rougher and scavenger flotation cells, with an estimated 5- and 15-minute retention times, respectively; 180-cubic foot first-cleaner-scavenger cells with 8-

minute retention; 100-cubic-foot second-cleaner cells with 5-minute retention; and 50-cubic-foot third-cleaner cells with 4-minute retention time. The middling would be thickened and reground in a ball mill before final cleaning, and both concentrate and tailing products thickened before leaving the system.

The bulk concentrate generated by the above procedure could then be separated into copper and nickel-copper concentrates using a flotation flowsheet such as the one diagrammed in Figure 6-2. Factors which are important for making this separation are reportedly as follows:

a. Control of temperature in the separators and cleaner-recleaners in the range of 80° to 95°F.

b. Close regulation of fresh water additions to cleaners and recleaners, in a ratio of 1:3 of cleaner-recleaner tailing by volume.

c. Additions of lime to maintain a level of about 1.4 pounds per ton of solution in the separators and 1.7 pounds per ton in the cleaners. This is combination with with about 0.04 pounds of NaCN per ton of original feed to the cleaner permit the successful depression of the pyrrhotite pentlandite,

d. Additions of sodium silicate of 1.5 pounds per ton in the separators facilitates the dispersion of the fine rock and chalcopyrite-cubanite.

Other than Minnesota's, the only major nickel deposit that contains much more copper than nickel is that of Noril'sk in Siberia. This operation produces several hundred million annual pounds of nickel from ore substantially higher in grade (double?) than that in Minnesota. The Norli'sk ore contains chalcopyrite, cubanite ($CuFe_2S_3$), pentlandite, and pyrrhotite, which are upgraded into a copper and nickel-copper concentrate. The nickel-copper concentrate typically assays 3.3% Ni. 3.0% Cue, 43% Fe, and 26% S (Goryachkin, 1978). Thus, the mineralogy of the Noril'sk deposit (which has been mined since 1940) is quite similar to that of the Minnesota Duluth Complex nickel deposit. Russian metallurgists have put substantial effort into increasing nickel recovery by processing nickeliferous pyrrhotite that normally either would down-grade the nickel-copper concentrate, or would go to tailings (Goryachkin, 1978; Matevich, 1976; Smagina, 1970; Vifian, 1968; Vororov, 1976). None of the Russian efforts at Noril'sk seem to be as viable economically as INCO's pyrrhotite treatment process in Copper Cliff (Boldt and Queneau, 1967). (INCO's iron-ore plant has been shut down in response to the current slump in demand for nickel and iron products.) Feed to INCO's plant was prepared by magnetic concentration of pyrrhotite from rougher flotation tailings generated during flotation of bulk Ni-Cu concentrate. Unfortunately, most of the pyrrhotite in Duluth Complex ore and Noril'sk ore is not strongly magnetic, and thus responds poorly to magnetic concentration.

Pyrrhotite treatment efforts at Noril'sk involve flotation separation of a nickeliferous pyrrhotite concentrate from Ni-Cu concentrate. The upgraded Ni-Cu concentrate is smelted. Base and precious metal values are separated from the pyrrhotite via a combination of pressure hydrometallurgy and flotation, then are sent to the nickel smelter. Iron is discarded as low-grade waste to tailings. Overall savings in recovery of non-ferrous metal values by secondary treatment of pyrrhotite is not known to this writer, who believes that the Noril'sk approach to pyrrhotite-iron treatment is uneconomic. Neither information nor resources are available to verify this belief. Nevertheless, technical developments at Noril'sk should be followed due to the similarity of the geology of this higher grade ore body to Duluth Complex.

Smelting Duluth Complex Concentrates

A detailed and authoritative description of conventional smelting procedures for producing in Minnesota a marketable white metal (matte) from Minnesota bulk Cu-Ni concentrate (Figure 6-1) is given by Lawver et al. (1975). (Note that the Lawver study (1975) was based on mining 25,000,000 tpy ore containing 0.45% Cu and 0.15% Ni.) Three applicable commercially-proven pyrometallurgical processes capable of meeting environmental requirements are: (1) fluid bed roasting followed by electric smelting; (2) Outokumpu flash smelting; or (3) INCO flash smelting. (Flash smelting entails combusting the sulfides as the particles of concentrate drop into the furnace.) All three processes are cost competitive, and would include an electric furnace for slag cleaning, as well as efficient SO_2-collection and sulfur-fixation systems (Figure 6-17). The white metal could be sold to a custom refiner, or refined locally. Recovery of copper and nickel from the bulk concentrate into the white metal would be about 98% and 95%, respectively. The above bulk-treatment approach is not preferred practice, regardless of where the white metal is refined.

For reasons discussed in the next paragraph, preferred production practice would involve flotation separation of bulk Cu-Ni concentrate into Cu and Ni-Cu concentrates. The copper concentrate would be smelted to produce blister copper anodes. Smelting of the nickel-copper concentrate would generate nickel matte and blister copper. The two smelters would be constructed side-by-side in northern Minnesota. The blister copper would be shipped to a domestic custom electrorefinery for conversion to high-purity

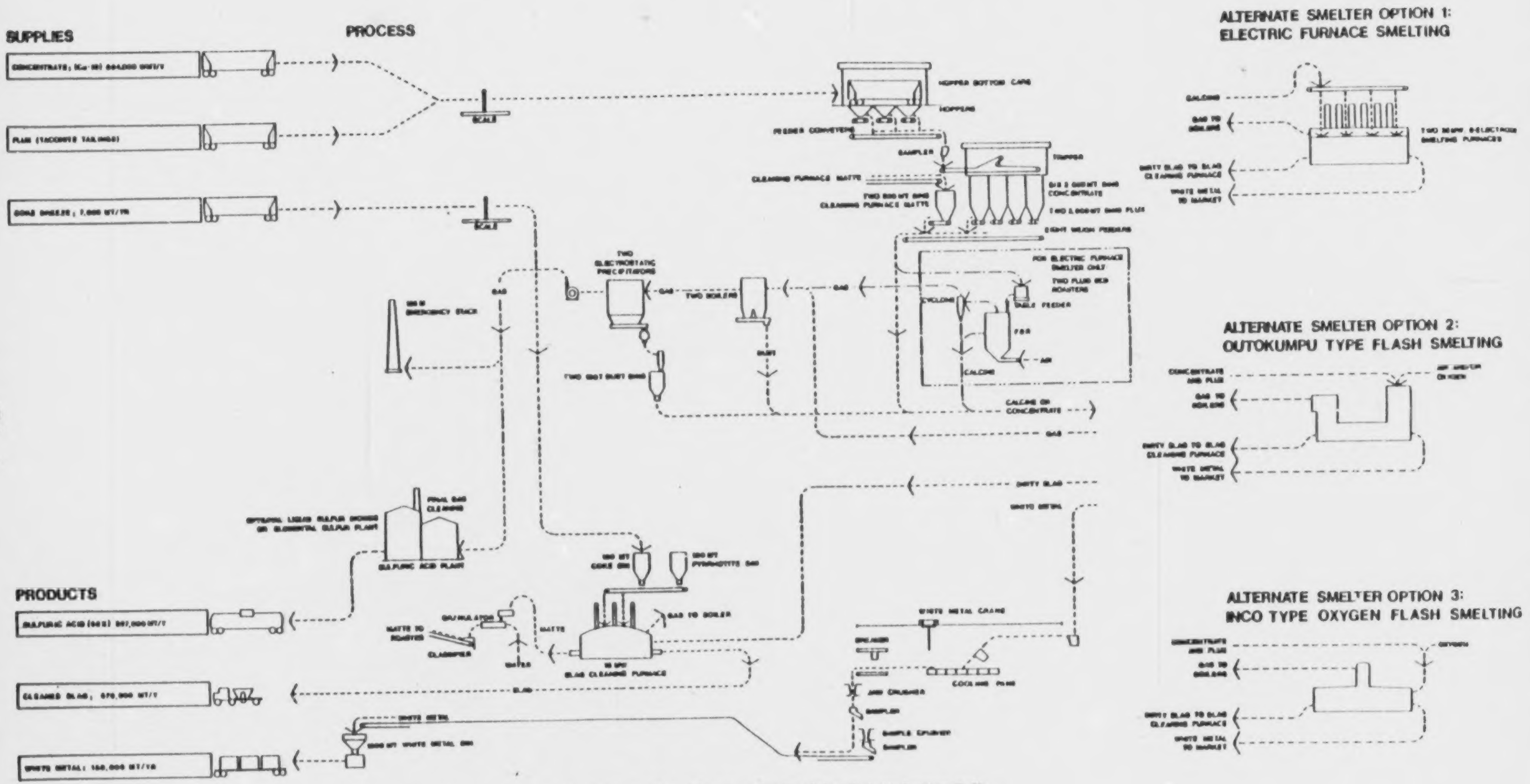

SUPPLIES
PROCESS
PRODUCTS
CONCENTRATE; (Cu-Ni)
FLUX (TACONITE TAILINGS)
COKE BREEZE; 7,000 MT/YR
SCALE
HOPPER BOTTOM CARS
HOPPERS
FEEDER CONVEYERS
SAMPLER
TRIPPER
CLEANING FURNACE MATTE
TWO 800 MT BINS CLEANING FURNACE MATTE
CONCENTRATE
FLUX
EIGHT WEIGH FEEDERS
FOR ELECTRIC FURNACE SMELTER ONLY
TWO FLUID BED ROASTERS
CYCLONE
TABLE FEEDER
FBR
AIR
CALCINE
CALCINE OR CONCENTRATE
GAS
DUST
TWO BOILERS
TWO ELECTROSTATIC PRECIPITATORS
EMERGENCY STACK
FINAL GAS CLEANING
OPTIONAL LIQUID SULFUR DIOXIDE OR ELEMENTAL SULFUR PLANT
SULFURIC ACID PLANT
COKE BIN
PYRRHOTITE BIN
GAS TO BOILER
GRANULATOR
MATTE TO ROASTERS
CLASSIFIER
WATER
MATTE
SLAG CLEANING FURNACE
SLAG
DIRTY SLAG
WHITE METAL
WHITE METAL CRANE
BREAKER
COOLING PANS
JAW CRUSHER
SAMPLER
SAMPLE CRUSHER
WHITE METAL BIN
SULFURIC ACID (98%)
CLEANED SLAG
WHITE METAL
ALTERNATE SMELTER OPTION 1:
ELECTRIC FURNACE SMELTING
CALCINE
GAS TO BOILERS
DIRTY SLAG TO SLAG CLEANING FURNACE
WHITE METAL TO MARKET
TWO SMELTING FURNACES
ALTERNATE SMELTER OPTION 2:
OUTOKUMPU TYPE FLASH SMELTING
CONCENTRATE AND FLUX
GAS TO BOILERS
DIRTY SLAG TO SLAG CLEANING FURNACE
WHITE METAL TO MARKET
ALTERNATE SMELTER OPTION 3:
INCO TYPE OXYGEN FLASH SMELTING
CONCENTRATE AND FLUX
OXYGEN
GAS TO BOILERS
DIRTY SLAG TO SLAG CLEANING FURNACE
WHITE METAL TO MARKET

FIGURE 6-17 Alternative Flow Diagrams for Smelting Duluth Gabbro Bulk Flotation Concentrate.

cathodes. The nickel matte would be shipped to AMAX's Port Nickel refinery for conversion to highly refined nickel and cobalt products plus a PGM concentrate. (Present capability at Port Nickel is limited to production of nickel and cobalt briquettes, plus copper cathodes.)

Separate treatment of copper and nickel-copper concentrates is preferable for two reasons. The first is that separation of most of the copper from the nickel by flotation is straightforward commercially-proven technology that is relatively inexpensive and does not generate additional tailings. The second reason is that the pyrometallurgical method to separate nickel from copper--"matte separation"--requires a nickel to copper ratio of at least one to work efficiently. Differential flotation provides a Ni-Cu concentrate that meets this ratio requirement. Matte separation involves slow-cooling the matte so that three phases segregate, one rich in copper (i.e., Cu_2S), one rich in nickel (i.e., $Ni3S_2$), and one rich in precious metals (i.e., Ni-Cu alloy). The segregated matte is then comminuted in a manner that produces a particulate mixture of the above phases. The copper and nickel sulfides are then separated by flotation, and the alloy by magnetic concentration. Note that an alternative to matte separation is chloride leaching followed by solvent extraction, as practiced by Falconbridge in Norway (Hougen, 1975).

If the Port Nickel refinery were upgraded to permit production of top-quality nickel, licensing INCO's IPC process (INCO pressure carbonyl) should be given careful consideration. Alternately a refinery utilizing carbonyl technology could be constructed in Minnesota. The overall processing costs are less, and the nickel product is exceptionally pure. Nickel matte and metallics produced by matte separation, plus Ni-Cu-Co-Fe matte from slag cleaning, would be smelted in a TBRC to produce a granulated (water-quenched) matte containing less than 5% sulfur. This matte would then be treated under pressure with carbon monoxide to convert most of the nickel and some of the iron to volatile carbonyl compounds (e.g., $Ni(CO)_4$, $Fe(CO)_5$). These vapors are selectively condensed and decomposed to produce high-purity nickel product plus ferronickel byproduct. The residue remaining after pressure carbonylation would be pressure-leached to recover copper, cobalt, and residual nickel values (Queneau, 1969). The remaining leach residue would contain the precious metals, which would be recovered as described in the write-up on recovery of platinum from domestic sources.

Recovery of Cobalt

The behavior of cobalt during smelting is midway between that of iron and nickel. Thus, much of the cobalt is lost to the slag along with the iron. About 70% of this cobalt could be recovered as a Cu-Ni-Co-Fe matte by cleaning the slags. Slag cleaning involves adding a reductant (e.g., coke) and a source of sulfur (e.g., pyrrhotite) to the molten slag in an electric furnace to convert the Cu-Ni-Co oxides to sulfides, which form a heavier matte product. During smelting of copper concentrate, all the cobalt reports to the slag. During nickel-copper concentrate smelting to produce a Ni-Cu matte containing about 1% iron, 75% of the cobalt is slagged. Low-iron Ni-Cu matte is necessary to attain a sharp separation between nickel and copper during the subsequent matte separation step (Matousek, 1982).

During matte separation, most of the cobalt follows the nickel into the Ni_3S_2 and alloy phases, from which cobalt recovery should be 90% to 95%. This high recovery assumes production of high-value-added nickel products (e.g., cathode, briquettes, carbonyl pellets), rather than nickel oxide. Production of nickel oxide from Minnesota matte has three disadvantages. First, oxide nickel is worth less per pound than cathode, briquettes, or pellets. Second, oxide nickel is produced by roasting (burning) the Ni_3S_2 product from matte separation, which locks the cobalt to the nickel for no sales credit. Third, production of oxide nickel requires minimizing the copper contamination of the Ni_3S_2 product of matte separation. This constraint necessitates that somewhat more nickel be allowed to report to the Cu_2S matte-separation fraction, which ultimately ends up in copper-converter slag.

A calculation of cobalt recovery from the Duluth Complex can therefore be summarized as follows. Given 100 units of cobalt in the ore, about 60 are lost to the mill tailings. Of the 40 remaining units, about ten units will be lost to cleaned slags and about three units will be lost during cobalt refining. Overall cobalt recovery may be on the order of 25 to 30%. Given an operation producing 100,000 tpy of copper from the Duluth Complex, associated cobalt recovery would thus be about 800 tpy, the exact figure depending on the orebody. This 800 tons depends on maximizing cobalt recovery during each processing step. The largest single cobalt loss (1800 tpy) is to the flotation tailings.

General Observations

Minimizing capital investment for developing the Duluth Complex would involve limiting the operation to the mine(s) and mill. The Cu and Ni-Cu concentrates could be shipped at reasonable cost to custom smelters.

Two papers by K. Natarayan et al. (1982) summarize biochemical leaching of Duluth Complex ore, flotation tailings, and Cu-Ni bulk concentrate. Their objectives were minimizing pollution from mine and tailing acid drainage, and selectively separating copper from the bulk concentrate. Although far more development work (e.g., five to ten years) remains to be completed, it is likely that bacteria-assisted leaching will find an application in whatever Duluth Complex process is eventually brought on line in Minnesota. Ideally, the tailings would be leached in place to recover nickel and cobalt values locked in the olivine.

Estimating the timing for startup of nickel production in Minnesota is a difficult problem. First, the existing depressed state of the primary metals industry must recover. Until unused production capacity comes back on line, there is little incentive to build new capacity. Furthermore, metal prices must recover. The economic calculations given in the 1975 Lawver report were based on $1.30/lb Cu and $3.50/lb Ni, converted to January, 1983 dollars. At these prices, the operation is marginally economic. Given 25% higher Cu-Ni prices, mining the Minnesota Duluth Complex for nonferrous metals should be profitable. Also, precious-metal values could be a significant factor in future Duluth Gabbro economic calculations. Time to bring the property on line would be on the order of five years. In this writer's opinion, it is not a question of whether, but when, copper-nickel mining operations will commence in Minnesota.

Hanna's Nickel Mountain Operation

Hanna Mining Company began producing nickel in 1954, and ceased operations in 1982. Also as noted in Chapter 5, moisture content of the ore is about 20% free water plus 8% water of crystallization. The minus 3-inch fraction of the crushed ore is delivered from the mine to the treatment plant via an 8300 feet long aerial tramline, which descends 2000 vertical feet off Nickel Mountain.

Three rotary kilns (10 x 100 ft) operate in parallel to dry the ore to about 4% moisture. As the ore tumbles down the kiln, the softer higher-grade material tends to break up, while the peridotite is little affected. The ore is then passed over a 5 x 12 ft double-deck screen which rejects plus one and one-quarter inch oversize to tailings (Figure 6-18). This rejection of low nickel oversize (approximately 0.7% Ni containing up to 10% of the nickel content of the ore) is common practice in processing laterite ore, in that the basement peridotite is not only low-grade, but also relatively hard. It is also high in magnesium content, which raises the ore's melting point. Intermediate-size material (minus one and one-half to plus three-eighths inch) from the screen is reduced to minus five-sixteenths inch by passing through two, 4 foot cone crushers operated in closed circuit (Coleman, 1960). The screen undersize drops into an air classifier, which separates out the minus 20-mesh fines. These fines are blended with ore dust recovered from the ore-drier off-gas. Fines are preheated in two, 10-hearth, 22 foot diameter calcining furnaces. The minus five-sixteenths inch crusher discharge is preheated in two parallel 9 foot by 250 foot rotary kiln calciners countercurrently fired by natural gas. Sawdust is added to all these furnaces, both to decrease fuel consumption and to reduce Fe_2O_3 to FeO. The resulting saving of ferrosilicon is substantial (Boldt and Queneau, 1967).

Smelting to Ferronickel

Reduction of the nickel and iron in the garnierite, which is soft compared to the peridotite rock waste, is carried out by a modified Ugine process. In Hanna's adaptation of this process, the final stages of reduction are completed with minus five-eighth inch ferrosilicon. Hot ore is gravity fed to three-phase, open top, hydraulically tilted melting furnaces (26 ft ID

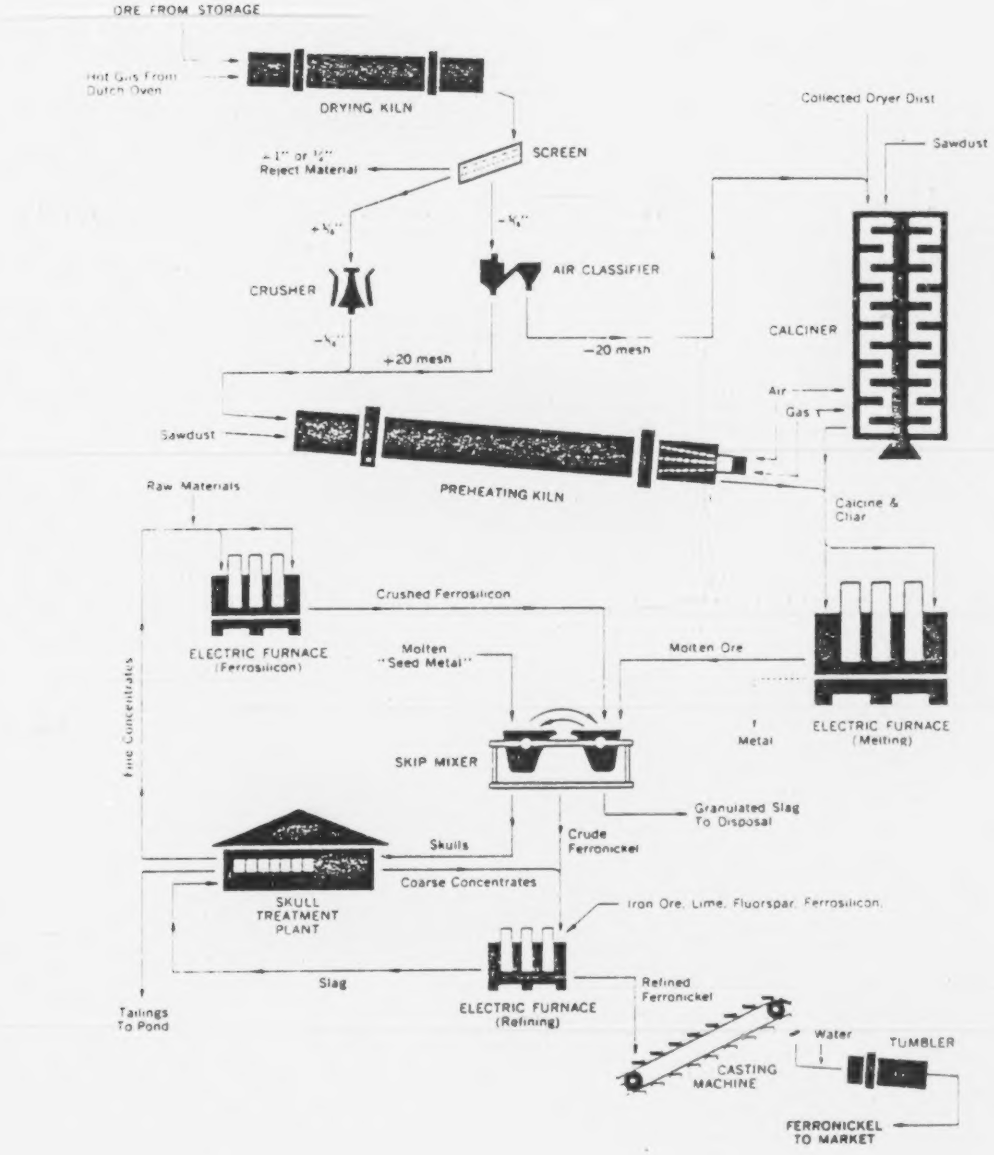

ORE FROM STORAGE
Hot Gas From Dutch Oven
DRYING KILN
SCREEN
+1" or ½" Reject Material
+⅛"
−⅛"
CRUSHER
AIR CLASSIFIER
−⅛"
+20 mesh
−20 mesh
Collected Dryer Dust
Sawdust
CALCINER
Air
Gas
Sawdust
PREHEATING KILN
Calcine & Char
Raw Materials
ELECTRIC FURNACE (Ferrosilicon)
Crushed Ferrosilicon
Molten "Seed Metal"
Molten Ore
Metal
ELECTRIC FURNACE (Melting)
SKIP MIXER
Granulated Slag To Disposal
Fine Concentrates
Skulls
Crude Ferronickel
Coarse Concentrates
SKULL TREATMENT PLANT
Iron Ore, Lime, Fluorspar, Ferrosilicon
Slag
Tailings To Pond
ELECTRIC FURNACE (Refining)
Refined Ferronickel
Water
TUMBLER
CASTING MACHINE
FERRONICKEL TO MARKET

FIGURE 6-18 Production of Ferronickel at Hanna's Operation in Riddle, Oregon (Boldt and Queneau, 1967).

by 10 ft deep). Since the ore is relatively nonconductive, the carbon electrodes are operated phase-to-phase, with electrodes spaced 80 inches center-to-center. Furnace temperature is about 1600^{o}C, and its capacity is 175 tons. The subsequent reduction process, as described by Coleman, is given below:

1. Approximately 15 tons of molten ore containing about 400 lb nickel are poured from one of the melting furnaces into a 280 ft^3 reaction ladle. This ladle is positioned in one side of the skip mixer. In the other side of the skip mixer, an identical ladle is waiting with about 11 tons of molten ferronickel.

2. The ladle containing ferronickel is raised by the skip mixer, and its contents poured rapidly into the ladle containing molten ore. At the same time, minus five-eighth inch ferrosilicon is added to the ladle. Reduction is very rapid, and considerable exothermic heat is released.

3. The empty ladle is set down and the full one, now containing the mixture of ore, seed metal (i.e., ferronickel) and ferrosilicon, is raised and rapidly poured into the empty ladle. Pouring from ladle to ladle continues for seven pours to assure good mixing.

4. The ladle in which the mixture remains after the above pours is allowed to stand for a minimum of three minutes to promote settling of metallic particles. A cast-steel slag ladle is then positioned in the skip mixer opposite the full reaction ladle. The full reaction ladle is raised and slag carefully decanted off the top of the metal.

5. The slag ladle is transported to the granulating position in the outer wall of the smelter building, and slag is granulated by high pressure

water jets so that the slag can be conveniently transported to the tailings dump. While the slag is being poured, the other melting furnace is being poured in preparation for the next reduction cycle, and the ladle is positioned in the skip mixer as soon as the slag ladle has been removed.

Each cycle takes about 15 minutes--some 11 minutes for mixing and reacting, 3 for settling, and 3 for removing slag and moving ladles. About 800 lb of ferronickel (48% Ni) is produced per cycle, adding that quantity to the seed metal. The cycles are repeated until the weight of seed metal has increased to 15 tons. Then 4 tons are removed (i.e., "thieved"). The thieved metal is sent in 8-ton lots to an electric refinery furnace. The ferronickel so produced is refined to remove phosphorous. This procedure involves adding iron ore to oxidize the phosphorous in the presence of a high-lime slag to form $Ca_4P_2O_9$, which is skimmed from the furnace. The ferronickel is finally cast into 28 pound pigs containing less than 0.05% each of P, S, C, and Cr. Nickel recovery from the melted ore is about 90%.

Skulls (frozen solids) from ladles, together with refining slag and spillage, comprise about 13% of the weight of molten ore. Ferronickel in these materials is recovered by crushing, magnetic separation, and screening. The recovered ferronickel is sent to the refining furnace (see Figure 6-18). Of 7000 tons per day of ore (about 85 tons of Ni) sent down the tramline, 8 tons of nickel were lost to the coarse screen fraction, and a slightly lesser amount was lost to smelter slag.

The economics of the furnace technology utilized at Hanna's Riddle operation depends on the availability of low-cost hydroelectric power, as well as reasonable ore-grade and nickel prices. The property's recent shutdown was the result of sharply escalating power costs, in combination with sharply decreasing nickel prices. Decreasing nickel grade of the remaining ore compounded the problem. Suggestions concerning a possible future for this plant are:

a. The plant was built originally as part of the Defense Production Act to establish a domestic source of nickel. If the decision were made to shut down the plant permanently, it might be "mothballed" rather than being scrapped;

b. The Riddle plant has valuable infrastructure, including power, natural gas, water, site development, tailings disposal facilities, process equipment, maintenance equipment, and offices. These facilities could be maintained, much the way Freeport Minerals mothballed Port Nickel through the 1960's; and

c. When required, the plant could serve as a processing facility for laterite nickel ores. The plant could be operated as is, or modified. Process upgrading could be INCO-type reduction smelting to matte, SLN-type reduction smelting to ferronickel, Moa Bay-type sulfuric acid pressure leaching, or Nicaro-type ammoniacal atmospheric leaching.

Ni-Co Form Nickeliferous Laterite by Hydrometallurgy

There are two commercially-proven processes for hydrometallurgical recovery of nickel and cobalt from laterite ores. One uses pressure leaching with sulfuric acid, as developed at Moa Bay (Carlson et al., 1961). The other involves atmospheric leaching of selectively reduced ore with aqueous ammonia, as developed by Caron (1950), and Nicaro (Boldt and Queneau, 1967). The Nicaro approach is shown in Figures 6-19 and -20. Figure 6-21 shows the modified Moa Bay process developed by Amax (Duyvesteyn, 1979; Kay, 1978). It

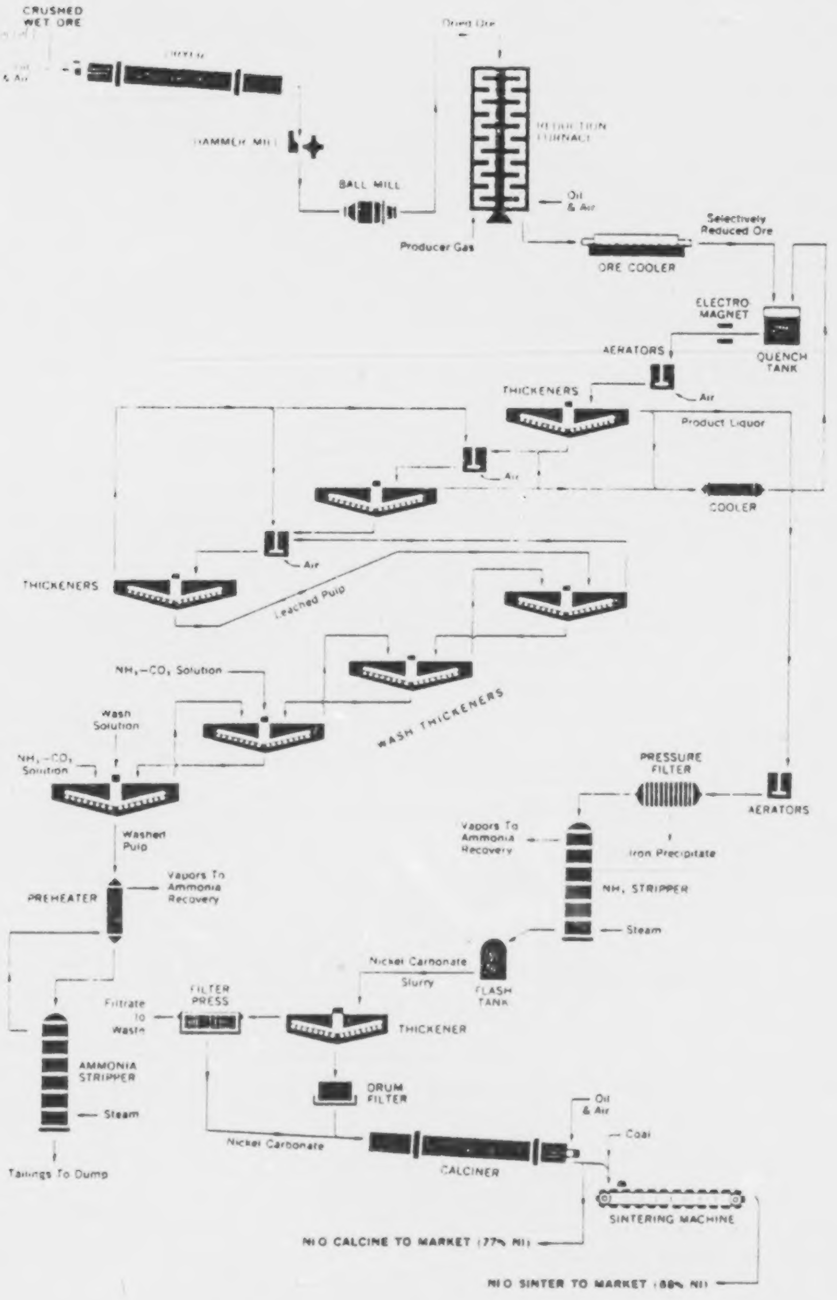

CRUSHED WET ORE
Oil & Air
DRYER
Dried Ore
HAMMER MILL
BALL MILL
REDUCTION FURNACE
Oil & Air
Producer Gas
Selectively Reduced Ore
ORE COOLER
ELECTRO MAGNET
AERATORS
QUENCH TANK
THICKENERS
Air
Product Liquor
Air
COOLER
Air
THICKENERS
Leached Pulp
NH_3-CO_2 Solution
Wash Solution
WASH THICKENERS
NH_3-CO_2 Solution
Washed Pulp
PRESSURE FILTER
AERATORS
Vapors To Ammonia Recovery
Iron Precipitate
NH_3 STRIPPER
Steam
PREHEATER
Vapors To Ammonia Recovery
Nickel Carbonate Slurry
FLASH TANK
Filtrate to Waste
FILTER PRESS
THICKENER
AMMONIA STRIPPER
Steam
DRUM FILTER
Oil & Air
Coal
Nickel Carbonate
CALCINER
Tailings To Dump
SINTERING MACHINE
NiO CALCINE TO MARKET (77% Ni)
NiO SINTER TO MARKET (88% Ni)

FIGURE 6-19 Nicaro Operation for Selective Reduction of Laterite Ore (Boldt and Queneau, 1967).

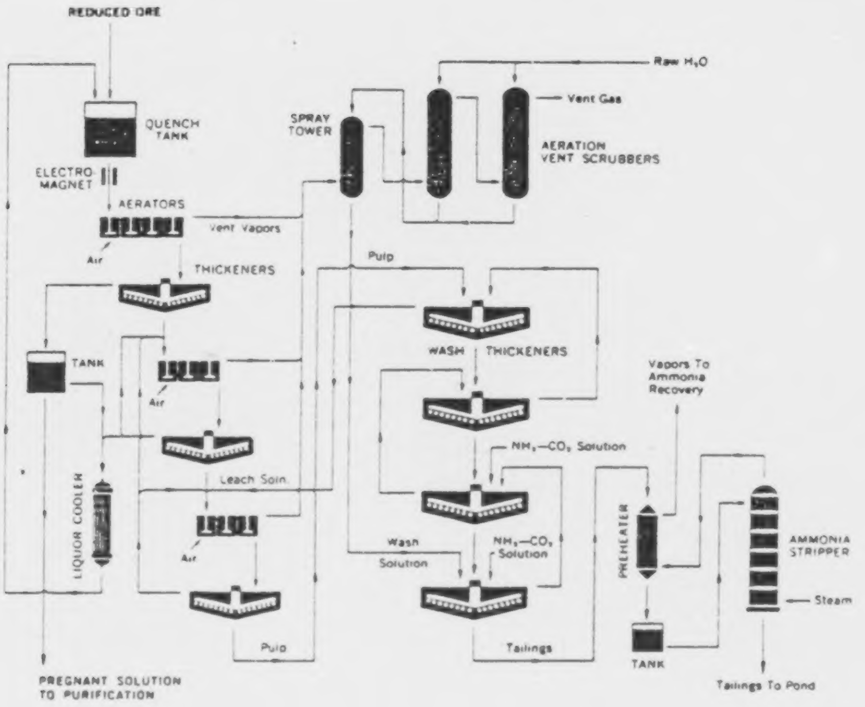

REDUCED ORE
Raw H_2O
Vent Gas
QUENCH TANK
SPRAY TOWER
AERATION VENT SCRUBBERS
ELECTRO-MAGNET
AERATORS
Vent Vapors
Air
THICKENERS
Pulp
TANK
Air
WASH THICKENERS
Vapors To Ammonia Recovery
NH_3—CO_2 Solution
LIQUOR COOLER
Leach Soln.
Air
Wash Solution
NH_3—CO_2 Solution
PREHEATER
AMMONIA STRIPPER
Steam
Pulp
Tailings
TANK
PREGNANT SOLUTION TO PURIFICATION
Tailings To Pond

FIGURE 6-20 Leaching Selectively-Reduced Laterite at Nicaro (Boldt and Queneau, 1967).

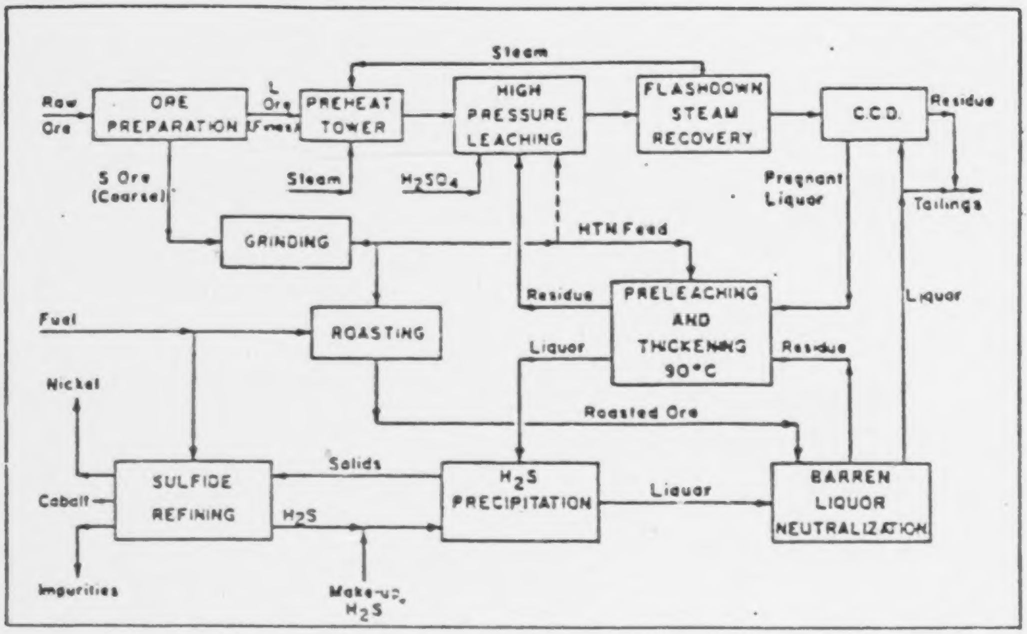

Steam
Raw Ore
ORE PREPARATION
L Ore (Fines)
PREHEAT TOWER
HIGH PRESSURE LEACHING
FLASHDOWN STEAM RECOVERY
C.C.D.
Residue
S Ore (Coarse)
Steam
H_2SO_4
Pregnant Liquor
Tailings
GRINDING
HTN Feed
Residue
PRELEACHING AND THICKENING 90°C
Liquor
Fuel
ROASTING
Liquor
Residue
Nickel
Roasted Ore
Solids
SULFIDE REFINING
H_2S PRECIPITATION
Liquor
BARREN LIQUOR NEUTRALIZATION
Cobalt
H_2S
Impurities
Make-up H_2S

FIGURE 6-21 Moa Bay Flowsheet for Acid Pressure Leaching of Laterite Ore, as Improved by Amax, Inc. (Kay, 1978).

applies to a broader range of laterite ores, attains higher metal recovery, and consumes less energy than does the original Moa Bay flowsheet. It is important to recognize that any "new" process advanced for leaching of lateritic ores (e.g., the Gasquet Mountain Project) must demonstrate superiority over the basic principles of the Nicaro and Moa Bay approaches. Ore-grades and types that can be processed hydrometallurgically at a given cost have been well established during the past 20 years by large-scale operations. The California Nickel project, as proposed for Gasquet Mountain ore, has not demonstrated that it measures up to the tonnage, ore-grade or technology requirements recognized as necessary for economic recovery of metal values from nickeliferous laterite ore.

Technical papers published in the 1979 AIME International Laterite Symposium give excellent profiles of world-class laterite ore bodies that can be contrasted with ore-grades reported at Gasqute Mountain (0.75% Ni; 0.07% Co). For example, Greenvale, Australia, laterite ores averages 1.6% Ni and 0.12% Co, supporting the mining of 2 million tpy ore. Reserves are estimated at over 40 million tons (Burger, 1979). It is treated by the Nicaro process. The Marinduque property in the Phillippines also uses the Nicaro process on ore averaging 1.2% Ni and 0.1% Co, supporting the mining of 3 million typ ore. Reserves are estimated at over 80 million tons (Colvin, 1979). The original Nicaro process (Cuba) treats ore assaying 1.5% Ni and 0.1% Co, the minable portions of which cover about 50 square miles. Cuba's Moa Bay ore contains about 1.4% Ni and 0.14% Co, with over 50 million tons of reserves reported in 1955 (Carlson, 1961). The magnesium content of the Moa Bay deposit is relatively low (i.e., about 1% as compared to 8% at Nicaro). Therefore sulfuric acid leaching, which dissolves magnesium, can be used rather than ammoniacal solution in which mangesium is relatively insoluble. Drier feed from INCO's Indonesian deposit assays 2.1% Ni, which is high enough in grade to justify reduction smelting. The mine is designed to support smelting of 2 million typ ore (Musu, 1979). INCO's ore from the Exmibal property in Guatemala (2.0% Ni) is also smelted. Design ore tonnage at Eximbal is 740,000 typ (Sopko, 1979). The Cerro Matoso deposit in Columbia reportedly contains 25 million tons of reserves averaging 2.6% nickel (Gomez, 1979). To be amenable to smelting, the ore should be high grade (over 2% Ni) and should have Mg/Si/Fe ratios suitable for satisfactory slag formation. Smelting to matte rather than ferronickel has the advantage of recovering at least a portion of the cobalt. Matte is also more amenable to production of a variety of high-purity nickel products.

PLATINUM

Introduction

The three most important domestic sources of primary platinum are Montana's Stillwater Complex, Minnesota's Duluth Complex, and Alaska's Goodnews Bay. Of the three, only Goodnews Bay has a history of commercial platinum production. However, substantial metallurgical work has been performed on both the Stillwater and Duluth deposits. INCO processed a 10,000 ton sample of Duluth Complex ore through its Creighton mill near Sudbury, Ontario. The extent of Stillwater-related process development is not known.

Placer deposits, such as Goodnews Bay, generate a high-grade platinum concentrate that can be sold directly to a precious-metal refinery for purification. Most of the world's platinum, however, is produced by concentrating PGM-rich nickel and copper sulfides. These sulfides are one of nature's natural collectors for platinum-group metals (PGM). It is fortuitous that froth flotation enables the metallurgist to economically separate sulfides from much larger volumes of waste rock. The PGM values, most of which are incorporated within the sulfide-mineral particles, concentrate along with the nickel and copper. Conventional Ni-Cu smelting and refining procedures are designed to produce a byproduct stream of PGM-rich solids relatively free of base metals. This PGM and precious metal concentrate then is delivered to a specialized processing facility designed to produce high-purity sponge or ingots for each of the component noble metals.

Conclusions

1. The key to the viability of PGM recovery from the Stillwater Complex is sufficient return from Pt-Pd sales to pay mining and beneficiation costs. Technology for mining, beneficiating, smelting and refining Stillwater Ni-CU-PGM concentrates is well established. Technical improvements that are most likely to significantly impact Stillwater platinum economics are in beneficiation and refining.

2. The most qualified North American plant for converting Stillwater concentrate to high-grade PGM concentrate suitable for direct refining to Pt-Pd sponge is that of INCO at Copper Cliff, Ontario. Also, a possibility would be the combined capabilities of Nickel, Louisiana and Cartaret, New Jersey, facilities.

3. Byproduct production of platinum and palladium from recovery of copper and nickel from Duluth Complex could amount to as much as 50,000 oz Pt/yr plus about 130,000 oz Pd/yr based on 200,000 typ copper production. The

dollar value of the associated base metals is about twenty-fold higher than the PGM values. Therefore, opportunities for technical breakthroughs affecting project viability lie with Cu-Ni rather than with PGM metallurgy.

5. There is one operating domestic production facility for platinum: Goodnews Bay. Projected annual platinum production capability reportedly could be 10,000 oz of heavy-metal placer concentrate. The grade of this concentrate is reportedly sufficiently high that it can be shipped directly to a platinum refiner, such as AMAX's Cartaret, New Jersey, refinery. Data on the extent of associated palladium values is not available.

Montana's Stillwater Complex

A field exploration program initiated in 1967 by the Johns-Manville Corp. resulted in discovery of a significant platinum-group-element ore body in the Stillwater Complex of Montana. The ore-bearing interval is generally continuous throughout the complex, and averages approximately 6 feet in thickness. Since this discovery, Manville Products Corp. has formed a joint venture partnership with Chevron U.S.A., called PGM Resources, through which considerable process development work has been completed. The Anaconda Minerals Company also holds claims in the area, and has formally proposed opening a 1000 tpd mine.

Mining proposals by PGM Resources would probably be at the rate of 2500 tpd ore. Expected dilution is 20%, based on the amount of wall rock expected to slough off from shrinkage stoping when the ore is drawn from the stopes.

Beneficiation

Typical grade of Stillwater ore is 0.45 oz/ton Pd + Pt, plus 0.15% Ni +

Cu. The ratios of Pd/Pt and Ni/Cu in both the ore and the concentrate are about 3.5/1 and 1.8/1, respectively (Conn, 1979). The PGM dollar value is thus 10 to 20 times greater than the base-metal values. The principal opaque minerals reporting to flotation concentrates prepared from bulk samples are pentlandite, chalcopyrite, and pyrrhotite, with minor sphalerite, galena, millerite, and marcasite. The principal PGM (i.e., platinum-group metals) minerals, in decreasing order of abundance, are braggite (Pt, Pd)S, vyotskite (PdS), moncheite ($PtTe_2$), and Pt-Fe alloys (Cabri, 1981).

PGM Resources has proposed a beneficiation plant similar to Impala's circuit in South Africa (Chadwick, 1980) (Figure 6-22). Two-stage crushing would reduce the ore to minus one-half inch, followed by closed-circuit ball milling (wet) to minus 200-mesh (i.e., the particles are less than 74 microns in diameter). Subsequent flotation should generate about 35 tpd (assuming 2500 tpd mining) of concentrate containing about 30 oz/ton Pd + Pt per ton at about 85% recovery. Other values include 0.3 oz Rh/to 0.5 oz Au/ton, 6.5% Ni, and 3.6% Cu. A likely refinery for this concentrate is Sudbury, Ontario (Berg, 1981). A 15% loss of PGM values to tailings in this process is primarily due to the need to produce a reasonable grade of concentrate. For any given material, even when using state-of-the-art concentration technology, there is an optimal level of recovery that is usually less than 100%. Higher recovery than optimum incurs greater cost than the incremental amount of recovered metal is worth. Optimum values may change in the future with increases in platinum prices.

The USBM recently published its bench-scale study of flotation concentration of metal values from PGM Resources' ore (Bennetts, 1981). The feed assayed 0.08 oz Pt/ton, 0.35 ox Pd/ton, 0.11% Ni, and 0.06% Cu. Analysis

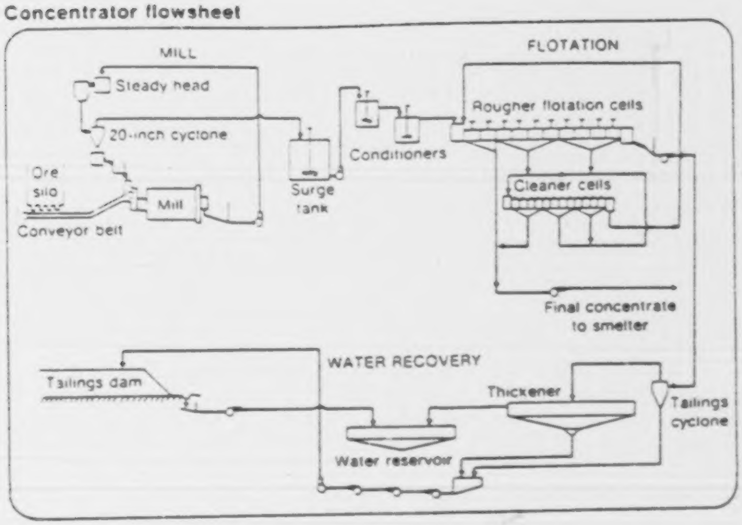

Concentrator flowsheet
MILL
Steady head
20-inch cyclone
Ore silo
Mill
Conveyor belt
Surge tank
Conditioners
FLOTATION
Rougher flotation cells
Cleaner cells
Final concentrate to smelter
WATER RECOVERY
Tailings dam
Thickener
Tailings cyclone
Water reservoir

FIGURE 6-22 Impala Platinum's Concentrator Flowsheet

of the cleaner of final stage concentrate is given below:

oz/ton		%	
Platinum	6.2	Copper	3.4
Palladium	21.0	Nickel	6.2
Rhodium	0.26	Iron	18.0
Iridium	0.08	Sulfur	16.0
Osmium	0.05	Silica	28.0
Gold	0.51	Magnesia	13.0
		Alumina	3.0

The concentrate also contained traces of lead (1200 ppm), cobalt (1100 ppm), phosphorous (980 ppm), and zinc (730 ppm). The ratio of concentration was about 75. Metal recoveries were 85% (Pt), 82% (Pd), 72% (Cu), and 67% (Ni). These results are comparable to those published by Johns-Manville (Conn, 1979). The preferred conditions for flotation, as developed by Bennetts, were as follows: grind, 100% minus 200-mesh, 67% minus 325-mesh; 40% solids; H_2SO_4, 22 lb/ton; AERO 404, 0.13 lb/ton, Dowfroth 250, 0.015 lb/ton; conditioning time, 10 min; and float time, 10 min. A somewhat different reagent combination has been suggested by Anaconda, as outlined in its Environmental Impact Statement (Montana Department of State Lands and U.S. Forest Service, 1982).

Smelting

The first step in upgrading the PGM concentrate, which would probably be done off-site at a smelter such as INCO's in Copper Cliff, is preparation of a furnace matte. Most of the silica, magnesia, and lime are slagged off. Laboratory preparation by the USBM of such a matte from Stillwater concentrate yielded material assaying 53% Fe, 32% S, 10.5% Ni, 7.5% Cu, 42 oz/ton Pd, and 15 oz/ton Pt (Baglin et al., 1982). Metal recoveries were 90% (Ni), 90% (Cu), 94% (Pd), and 94% (Pt). Commercially, the USBM preparation could be carried out with substantially higher recovery, such as is done at Rustenburg (Mostert, 1973), and Impala (Plasket et al., 1976), as shown in Figure 6-23. The furnace matte would then be converted to slag off most of the iron, and the matte so produced prepared for further refining. Recovery of platinum and palladium from Ni-Cu concentrate to Pt-Pd product, including recycle of intermediates, should be not less than 98%. Key to high recovery is capability to recycle and/or clean all process wastes, particularly the slags.

The tonnage of sulfide concentrate generated by a milling operation at the Stillwater Complex would be 30 to 40 tpd (assuming a 2500 tpd operation) containing about 30 oz/ton of Pd + Pt. Substantial savings in capital would result from shipping this concentrate to an existing smelter, such as INCO's operation in the Sudbury district of Ontario, or possibly that of Falconbridge. Precious-metal recovery would also be maximized due to availability of furnaces to handle recycled slags. How much of this recovery increase might benefit the Montana operation would depend on the nature of the contracts with INCO. A description of the handling of precious metals at INCO's Copper Cliff refinery complex is given by Barnes et al. (1982) (Figure 6-24). The contained nickel and copper in Stillwater concentrate would be less than 1% of that processed at Copper Cliff. No process compatibility problems are foreseen. Stillwater matte would probably be added to a furnace-matte converter, i.e., to the box labeled "smelting" in Figure 6-24. Nickel-copper matte produced in the smelting step is separated into Ni_3S_2, Cu_2S, and an alpha-nickel metallic fraction (approximately 200 tpd) via matte separation (Sproule et al., 1960). Most of the PGM go into the metallic fraction, which is then smelted in a TBRC (i.e., top-blown rotary converter) to generate low-sulfur activated matte for INCO pressure carbonyl (IPC) refining (Head et al., 1976). During carbonylation, most of the nickel and iron are volatilized,

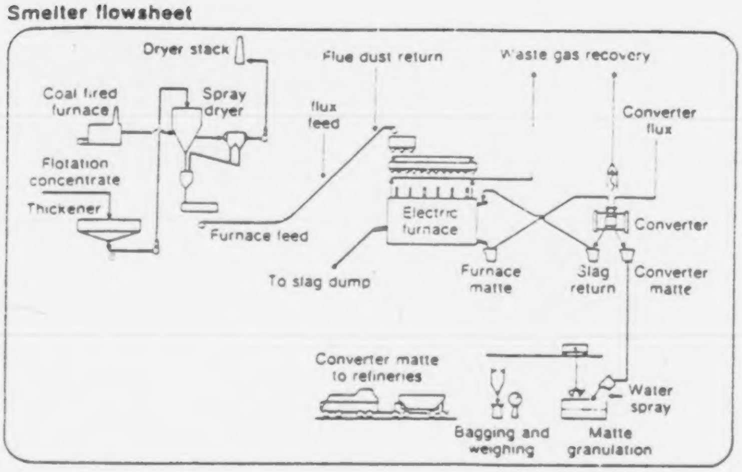

Smelter flowsheet
Dryer stack
Flue dust return
Waste gas recovery
Coal fired furnace
Spray dryer
flux feed
Converter flux
Flotation concentrate
Thickener
Electric furnace
Converter
Furnace feed
To slag dump
Furnace matte
Slag return
Converter matte
Converter matte to refineries
Water spray
Bagging and weighing
Matte granulation

FIGURE 6-23 Impala Platinum's Smelter Flowsheet

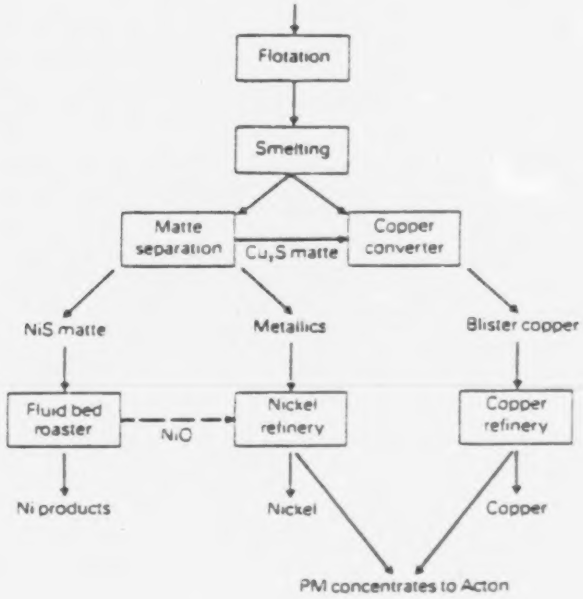

Flotation
Smelting
Matte separation
Cu$_2$S matte
Copper converter
NiS matte
Metallics
Blister copper
Fluid bed roaster
NiO
Nickel refinery
Copper refinery
Ni products
Nickel
Copper
PM concentrates to Acton

FIGURE 6-24 INCO's Copper Cliff Refining Complex.

leaving about 70 tpd of residue. High purity nickel pellets and ferronickel powder are produced from the condensed carbonyl compounds. The non-volatile residue is pressure-leached to remove nickel, cobalt, iron, and arsenic (Opratko et al., 1981). A final atmospheric leach is used to remove residual iron, which prepares the PGM-rich residue for treatment at INCO's Acton refinery in London.

Top-blown rotary converting has the advantage of volatilizing substantial quantities of Pb, As, Sb, Sn, Te, and Se, all of which otherwise tend to report to the PGM concentrate. The volatilized metals are thoroughly scrubbed from the TBRC off-gases, not only for environmental reasons, but also to recover entrained precious metals. It is this writer's understanding that the TBRC operation presents no foreseeable legal or regulatory difficulties. Even when using TBRC smelting, excessive quantities of lead and selenium may end up in the residue generated by the subsequent pressure leaching process. (Even though the total weight of lead and selenium are small, their concentration becomes significant as they concentrate along with the precious metals.) A sulfation roast, followed by leaching, may be necessary before final refining for recovery of high-purity platinum and palladium sponge. Such a roasting step is incorporated into a refining flowsheet published by Johns-Manville (Lake et al., 1978). Processing Stillwater matte at INCO is not expected to require process modifications.

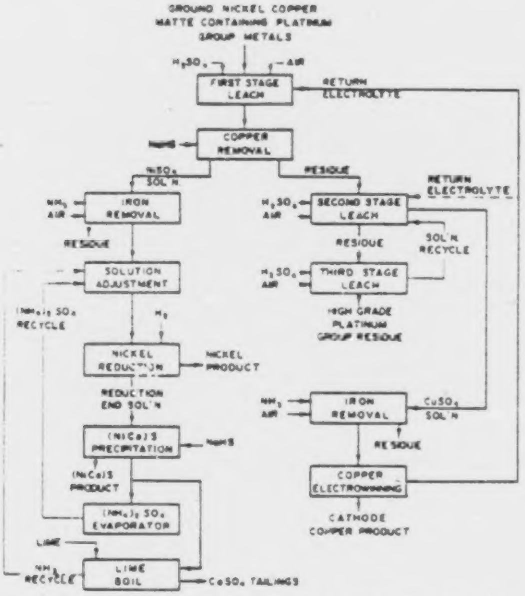
An alternative method for processing PGM-rich converter matte, as evidenced by the flowsheets chosen by commercial refiners, is via direct leaching (i.e., no carbonyl treatment). Impala Platinum (South Africa), the second largest producer of platinum in the western world, uses three stages of sulfuric acid pressure leaching to reject nonferrous metals (Plasket-

Romanchuk, 1977), as shown in Figure 6-25. Additional details of Impala's operation are given in papers by Ireland (1974) and Plasket-Wortley (1977). Matthey Rustenburg (South Africa) also uses a sulfate-based pressure leach to concentrate PGM values (Clemente et al., 1980). Falconbridge treats high-grade precious-metal mattes in Norway with aqueous $HCl-Cl_2$, producing a precious metal concentrate having a bulk no greater than 1% of the original matte (Hougen, et al., 1975); see Figure 6-26.

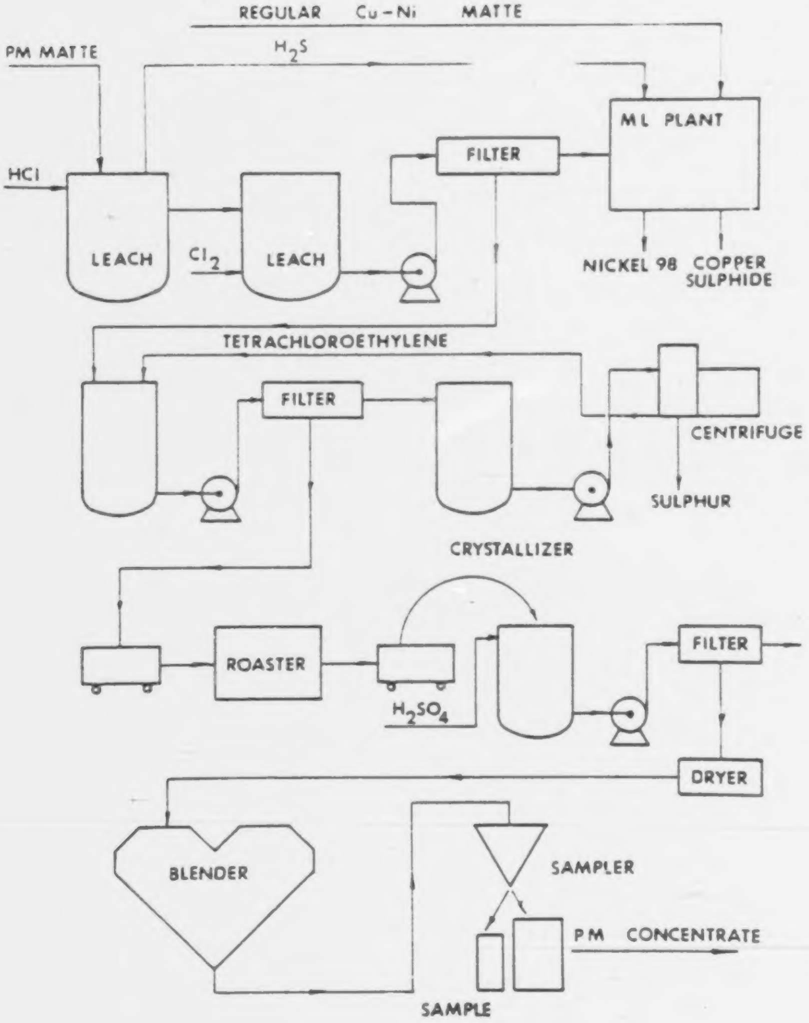
A typical conventional refinery mode for producing high-purity metals from PGM concentrates was detailed by Clements in 1962. These processes relied on skillful but cumbersome manipulation of the inorganic chemistry of the PGM metals (Stern, 1981; Westland, 1981). Recent innovations in solvent extraction (SX) and ion-exchange (IX) have simplified precious-metal refining, deleting many of the recycle streams and minimizing processing time. The PGM residue is dissolved completely, rather than selectively. The metals are then extracted selectively and sequentially from the total leach liquor. The chemistry of these SX and IX processes is described in papers by Charlesworth (1981), Cleare et al., (1979), and Edwards (1979, 1976). Matthey Rustenburg and Lonrho Refining (South Africa) are making extensive use of SX and IX. Flett (1983) gives an excellent review of these processes. Additional details are provided by Edwards (1979) and Berry (1979). Figure 6-27 gives a block diagram of INCO's new PGM-refinery process at Acton (Barnes, 1982), which was the first major PGM refinery to incorporate SX into its process (early 1970's).

In summary, the only integrated smelter in North America for preparing a PGM concentrate from Ni-Cu furnace matte is INCO's Copper Cliff, Ontario, operation. A possibility is AMAX's Port Nickel refinery, which would

FIGURE 6-25


GROUND NICKEL COPPER MATTE CONTAINING PLATINUM GROUP METALS
H_2SO_4
AIR
FIRST STAGE LEACH
RETURN ELECTROLYTE
NaHS
COPPER REMOVAL
$NiSO_4$ SOL'N
RESIDUE
NH_3
AIR
IRON REMOVAL
RESIDUE
SOLUTION ADJUSTMENT
$(NH_4)_2SO_4$ RECYCLE
H_2
NICKEL REDUCTION
NICKEL PRODUCT
REDUCTION END SOL'N
(NiCo)S PRECIPITATION
NaHS
(NiCo)S PRODUCT
$(NH_4)_2SO_4$ EVAPORATOR
LIME
NH_3 RECYCLE
LIME BOIL
$CaSO_4$ TAILINGS
RETURN ELECTROLYTE
H_2SO_4
AIR
SECOND STAGE LEACH
RESIDUE
SOL'N RECYCLE
H_2SO_4
AIR
THIRD STAGE LEACH
HIGH GRADE PLATINUM GROUP RESIDUE
NH_3
AIR
IRON REMOVAL
$CuSO_4$ SOL'N
RESIDUE
COPPER ELECTROWINNING
CATHODE COPPER PRODUCT

Simplified flow sheet. Recovery of nickel and copper from high-grade matte.



REGULAR Cu-Ni MATTE
PM MATTE
H_2S
HCl
ML PLANT
FILTER
LEACH
Cl_2
LEACH
NICKEL 98
COPPER SULPHIDE
TETRACHLOROETHYLENE
FILTER
CENTRIFUGE
SULPHUR
CRYSTALLIZER
ROASTER
H_2SO_4
FILTER
DRYER
BLENDER
SAMPLER
PM CONCENTRATE
SAMPLE

FIGURE 6-26 Schematic flowsheet of the Falconbridge Nikkelverk A/S Plant for Treating High-Grade PGM Matte.

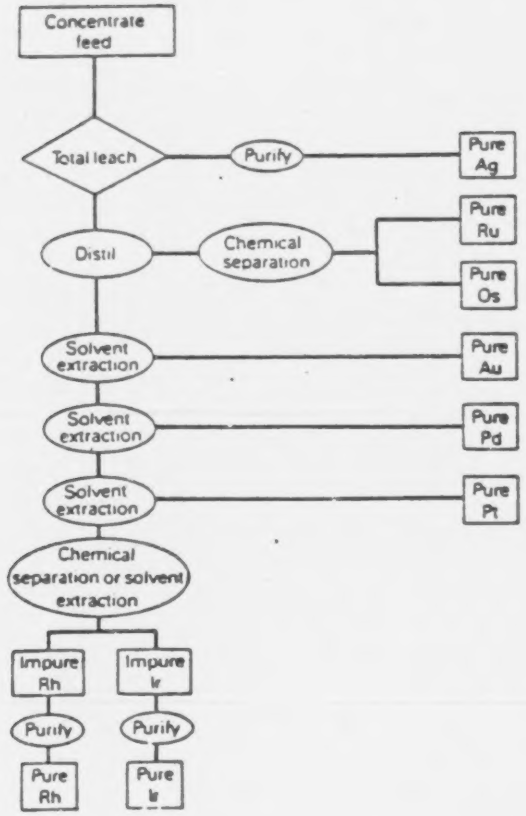

Concentrate feed
Total leach
Purify
Pure Ag
Distil
Chemical separation
Pure Ru
Pure Os
Solvent extraction
Pure Au
Solvent extraction
Pure Pd
Solvent extraction
Pure Pt
Chemical separation or solvent extraction
Impure Rh
Impure Ir
Purify
Purify
Pure Rh
Pure Ir

FIGURE 6-27 INCO's Solvent-Extraction Process for Separation and Purification of Precious Metals

concentrate the PGM values into a high-iron residue. This residue is presently shipped to AMAX's secondary-copper smelter in Cartaret, New Jersey, for smelting into a PGM concentrate, which is refined on-site. A third option is shipping the Stillwater concentrate to refineries in South Africa or England, Belgium, or West Germany. The most expedient and probably the most economic option is smelting at Copper Cliff. The best interests of the United States could be served by upgrading the Port Nickel refinery so that its capability to economically produce a PGM concentrate was at least a close second to INCO's. The third option, shipping abroad, is feasible but appears undesirable, both economically and politically.

Once a high-grade PGM concentrate has been prepared, it is an item of commerce that could be sold to any number of refineries including those assoicated with domestic copper refineries. The sale price will probably reflect use of the best technology for separation and purification of the component metals. Modern refineries handling large volumes of PGM values (e.g., Acton, Matthey, Rustenburg, Lonrho) should make the most profit from a given sales contract.

It is quite clear that the economic viability of recovering platinum from the Stillwater Complex reduces down to the viability of the mining and beneficiation steps. There is no question that the mining and beneficiation are technically viable. At what Pt-Pd prices these operations become economically viable is not known to this writer. Nor are the data available to the writer from which such a judgment could be made. One target for R&D dollars would be on beneficiation and refining technology. Particularly significant would be increasing PGM recovery while maintaining constant overall processing cost. However, improving the state-of-the-art of mining and beneficiation technology is likely to be more difficult then improving smelting and refining practice.

Minnesota's Duluth Complex

The matte generated by smelting bulk flotation concentrate produced from Minnesota Duluth Complex ore would assay approximately 62% Cu, 13% Ni, 21% S, 4% Fe, 5 oz/ton Ag, 0.15 oz/ton Au, 0.44 oz/ton Pd, and 0.19 oz/ton Pt (Lawver, 1975). Reference should be made to the section entitled, "Nickel" for additional discussion of processing these ores. This matte could be sold to an existing refinery, such as INCO's Copper Cliff facility, or to AMAX's Port Nickel refinery provided its processing capabilities were upgraded. At INCO, the matte could be blended with Ni-Cu matte and processed via matte separation, where the PGM values would follow the alpha-nickel metallics into the carbonyl plant. Refer back to the earlier discussion of smelting operations for the Stillwater PGM matte, and particularly to Figure 6-3. In fact, a grass-roots smelter ought to be constructed in northern Minnesota to process Ni-Cu and Cu concentrate produced by differential flotation, as described in the section on nickel recovery from domestic sources.

Differential flotation generates a nickel concentrate having a fairly low ratio of Cu/Ni (e.g., about 1/1). Such a concentrate, after converting to Ni-Cu matte, could be refined at a refinery constructed in Minnesota. Alternatively, the matte could be treated at INCO, or by AMAX's Port Nickel refinery in Louisiana using the existing flowsheet at Port Nickel. At Port Nickel, the PGM values would be concentrated in a high-iron pressure-leach residue, as described earlier. Note that, in order to minimize problems of residue recycle in Port Nickel's total pressure-leach circuit, matte iron content should be less than 3%, and preferably between 1% and 2% Fe. This lower iron content, as compared to 4% Fe, would result in 75% of the cobalt in the matte reporting to the smelter converter slag. Note that INCO's matte separation process also requires low-iron feed. A solution to this cobalt recovery problem is presented in the write-up on process metallurgy for nickel recovery from domestic sources.

The PGM and precious metal content of Duluth Complex ore can make up about 10% of the dollar value, depending on metal prices. This revenue is important to the economic viability of Cu-Ni production in Minnesota, and is not merely "icing on the cake."

Alaska's Goodnews Bay

The Goodnews Bay Mining Company operated from 1937 to 1976, until excessive overburden caused operations to cease. Dredging recovered a particulate heavy-metal concentrate, much of which was sent to Engelhard in New Jersey for refining. Refining of high-grade platinum concentrates has a well-established technology, the cost of which is usually small compared to the cost of mining and concentration. In 1980, the deposit was purchased by Hanson Properties, Inc. of Spokane. Dredging on a limited scale was reinitiated in 1981.

Milling of the platiniferous gravels is carried out by gravity methods. The gravels are first sized by a trommel. Fines between three-sixteenths and five-eighth inch are screened in the first section of the trommel, then sent to riffle tables followed by jigs and sluices. The mid-section of the trommel screens out intermediate-sized material, which is subsequently jigged and sluiced. Plus two and one-half inch solids discharged at the end of the trommel are diverted to a washing/grinding unit called a "mud hog." This unit breaks up platiniferous clay materials. The slimes (i.e., fine-grained material) of the "mud hog" are returned to the trommel intake for processing by the jigs and sluices. Cleanup of all concentrated material is completed at the camp using 4 x 8 ft Wilfley tables (Carnes, 1981).

Nickel

Boldt, J. R. and Queneau, P.E., The Winning of Nickel, Longmont Publishers, Toronto, 1967.

Caron, M. H., "Ammonia Leaching of Ni-Co Ores," JOM, p. 67, Jan. 1950.

Carlson, E. T. and C. S. Simons, "Pressure Leaching Nickeliferous Laterites with Sulfuric Acid," Extraction Metallurgy of Copper, Nickel, and Cobalt, p. 363, edited by P.E. Queneau, Interscience Publishers, N.Y., Longmont Publications, Toronto, 1961.

Coleman, E. E., "Production of Ferronickel at Riddle, Oregon," Extractive Metallurgy of Copper Nickel, and Cobalt, Interscience Publishers, p. 263, Edited by P.E. Queneau, Intersicence Publishers, N.Y..

Dames & Moore, "Revised Operational Plan, California Nickel Corp., Gasquet Mountain," 1982.

Duyvesteyn, W. P. C. et al., "Omniverous Process for Laterite Deposits," AIME Internat. Laterite Symp., p. 527, edited by D.J.I. Evans et al., New Orleans, LA, 1979.

Goeller, L. A. et al., "The Recovery of Cobalt: Metallurgy and Uses," Int. Conf. Cobalt, Brussells, 1981.

Goryachkin, V.I. et al., "Developing a Technology for Hydrometallurgical Beneficiation of Noril'sk Pyrrhotite Concentrate," Tsvetnye Metally, p. 1, Jan. 1978.

Haas, L.A. et al., "Autoclave Leaching of Copper-Nickel Matte Produced from Duluth Gabbro," USBM RI 8340, 1979.

Haas, L. A. et al., "Low-Pressure Leaching of Duluth Complex Matte," USBM RI 8522, 1981.

Head, M. D. et al., "Nickel Refining by the TBRC Smelting and Pressure Carbonyl Route," Annual AIME Meeting, Las Vegas, 1976.

Iwasaki, I. et al., "By-Product Recovery from Copper-Nickel Bearing Duluth Gabbro," Resources and Conservation, Vol. 9, p. 105, 1982.

Joyce, F. E., Jr., "Extraction of Copper and Nickel from the Duluth Gabbro Complex by Selective High Temperature Sulfatization," USBM RI 7475, 1980.

Kay, H. et al., "The Amax Acid-Leach Process for Oxide Nickel Ores," AIME-TMS Paper A78-36, 1978.

Kukura, M. E. et al., "Development of the UOP Process for Oxide Silicate Ores of Nickel and Cobalt," AIME Intl. Laterite Symp., New Orleans, 1979.

Lawver, J. E. et al., "Mineral Beneficiation Studies and an Economic Evaluation of Minnesota Copper-Nickel Deposit from the Duluth Gabbro," USBM

Grant No. G0144109, Mineral Resources Research Center, University of Minnesota, 1975.

Lewis, J. D., "Ely Spruce Prospect," USBM Deposit Summary Report, 1978.

Lewis, J. D., "Babbitt Lake:Minnamax," USBM Deposit Summary Report, 1980.

Matevich, T. N. et al., "Hydrometallurgical Processing of Pyrrhotite Concentrates at Noril'sk," Tsvet. Metal., Vol 17(7), p. 8, 1976.

Matousek, J. W., "The Pyrometallurgical Mining of Cobalt," CIM Bulletin, p. 121, Dec. 1982.

Matthews, N. A. and Sibley, F. S., "Nickel," Minerals, Facts, and Problems, USBM Bull. 671, p. 611, 1980.

Mineral Sciences Div., UOP, Inc., "New Procedure for Recovering Nickel and Cobalt from Western Laterites, Economic Feasibility," 1982.

Mussler, R. E. et al., "Electrowinning Nickel and Cobalt from Domestic Laterite Processing," USBM RI 8604, 1982.

Natarajan, K. A. et al., "Environmental Leaching Behaviour of Copper-Nickel Bearing Duluth Gabbro and Flotation Tailings," Mineral Resources Research Center, University of Minnesota, to be published in Hydrometallurgy 1982.

Natarajan, K. A. et al., "Microbial Aspects of Hydrometallurgical Processing and Environmental Control of Copper/Nickel-Bearing Duluth Gabbro," Proc. of 43rd Annual Mining Symp., University of Minnesota, Minneapolis, 1982.

Nilsen, D. N. et al., "Solvent Extraction of Cobalt from Laterite-Ammoniacal Leach Liquors," USBM RI 8419, 1980.

Queneau, P.E. et al., "Some Novel Aspets of the Pyrometallurgy and Vapometallurgy of Nickel," JOM, July 1969.

Queneau, P.E. et al., "U.S. Patent 2,944,883 (1960).

Reid, Kenneth J. et al., "Copper-Nickel Pilot Plant Studies at the Mineral Resources Research Center, Proc. of the 40th Annual Mining Symp., University of Minnesota, Minneapolis, 1979.

Richardson, J. Mark, "The Recovery of Metal Values from Nickel-bearing Laterite Ores by Reductive Roast/Ammonia Leach Technology," Process and Fundamental Considerations of Selected Hydrometallurgical Systems, p. 37, 1981.

Ridenour, J. et al., "Nickel Mountain," USBM Deposit Summary Report, 1978.

Schluter, R. B. et al., "Continuous Pilot Plant Testing Confirms Flotatability of Duluth Complex Sulfides," E&MJ, p. 172, April 1976.

Schluter, R. B. et al., "Flotation Responses of Two Duluth Complex Copper-Nickel Ores," USBM RI 8509, 1980.

Schluter, Robert B., "Separation of Copper-Nickel Mattes from Duluth Gabbro Concentrates by Flotation and Magnetic Methods," USBM RI 8646, 1982.

Sibley, S. F., "Nickel," USBM Minerals Yearbook, p. 585, 1980.

Siemens, R. E. et al., Recovery of Nickel and Cobalt from Low-Grade Domestic Laterites," USBM RI 8027, 1975.

Smagina, E. N., "Ore Processing in Strongly Magnetic Fields for Treating Pyrrhotite-containing Ni-Cu Concentrates," Tsvet, Metal., Vol. 11(7), p. 84, 1970.

Smith, K. A. et al., "Chlorination Route to Copper, Nickel, and Elemental Sulfur from Duluth Gabbro Sulfides," Proc. of 37th Annual Mining Symp., University of Minnesota, Minneapolis, 1976.

Sobol, S. I., "Use of Autoclaves in Conversion of Copper-Nickel Concentrates," Tsvet, Metal., Vol. 32(2), p. 34, 1979.

Sopko, Michael D., "The Exmibal Nickel Project," AIME Internat. Laterite Symp., p. 272, edited by D.J.I. Evans, et al., New Orleans, LA, 1979.

Toomver, T. T., "Developments of INCO's Selective Reduction Smelting Process for Nickel Laterite Ores," AIME Internat. Laterite Symp., p. 252, edited by D.J.I. Evans et al., New Orleans, LA, 1979.

Tseiner, V. M. et al., "Selecting Technology for Treating Cu-Ni Sulfide Concentrates," Tsvet. Metal., Vol. 13(10), p. 8, 1972.

Vifian, A. R. et al., "Mineralogical and Beneficiation Studies of the Copper-Nickel Bearing Duluth Gabbro," Proc. 29th Ann. Min. Symp. and 41st Ann. Meeting of Minn. Sec., AIME, Duluth, Minn., p. 183, 1968..pn204

Voronov, A. B. et al., "Hydrometallurgical Processing of Pyrrhotite-bearing Cu-Ni Concentrates," Tsvet. Metal., Vol. 17(1), p. 24, 1976.

Hough, L.R. et al., "Recovery of Nickel, Copper, and Precious-Metal Concentrate from High-Grade Precious-Metal Mattes," JOM, p. 6, May 1975.

Queneau, P.E., O'Neill, C.E., Illis, A., and Warner, J.S., "Some Novel Aspects of the Pyrometallurgy and Vapormetallurgy of Nickel," JOM, July 1969.

Berger, P.A., The Greenvale Nickel Laterite Ore Body," AIME International Laterite Symp., p. 24, edited by D.J.I. Evans et al., New Orleans 1979.

Colvin, N. et al., "The Marinduque Surigas Nickel Refinery," AIME International Laterite Symp., p. 346, edited by D.J.I. Evans et al., New Orleans, 1979.

Musu, R. et al., "P.T. Inco's Indonesia Nickel Project," AIME International Laterite Symp., p. 300, edited by D.J.I. Evans et al., New Orleans 1979.

Gomez, R. et al., "The Cerro Matoso Nickel Project," AIME International Laterite Symp., p. 412, edited by D.J.I. Eans et al., New Orleans 1979.

Platinum

Anon., "Anaconda Begins Permit Process for Stillwater Complex," E&MJ, p. 11, Sept., 1981.

Anon., "Spokane Industrialist Reopens only U.S. Platinum Mine," E&MJ, p. 41, July, 1981.

Anstett, T. F. et al., "Platinum Availability - Market Economy Countries," USBM IC 8897, 1982.

Baglin, E. G. et al., "Recovery of Platinum-Group Metals from Stillwater Complex, MT, Flotation Concentrates by Matte Smelting and Leaching," USBM RI 8717, 1982.

Barnes, J. E. et al., "Acton Precious Metal Refinery," Chemistry and Industry, p. 151, March, 1982.

Bennetts, J., "Preparation of Pt-Pd Flotation Concentrate from Stillwater Ore," USBM RI 8500, 1981.

Bert, A. W., "The Stillwater PGM Resources," USBM Deposit Summary Report, 1981.

Berry, R. I., "Refining Precious Metals," Chem. Eng., p. 90, June 18, 1979.

Cabri, L. J., "Relationship of Mineralogy to the Recovery of Platinum-Group Elements from Ores," CIM Special Vol. 23, p. 235, 1981.

Carnes, R. D., "Platinum Group Metals, Salmon River, Alaska," USBM Deposit Summary Report, 1981.

Chadwick, J. R., "Impala Platinum Preparing to Pass 1,000,000 Ounces Yearly," Mining World, 33(12), p. 42, 1980.

Charlesworth, P., "Separating the Platinum Group Metals by Liquid-Liquid Extraction," Platinum Met. Rev., Vol. 25(3), p. 106, 1981.

Cleare, M. J. et al., "Solvent Extraction in Platinum Group Metal Processing," J. Chem. Tech. Biotechnol., Vol. 29, p. 210, 1979.

Clemente, D. De J., "MRR's Cobalt Refining Operation — South Africa's First," CIM Hydromet. Meeting, Edmonton, 1980.

Clements, F. S., "Twenty-Five Years' Progress in Platinum Metals Refining," The Industrial Chemist, p. 345, July, 1962.

Conn, H. K., "The Johns-Manville Platinum-Palladium Prospect, Stillwater Complex, MT, USA," Canadian Mineralogist, Vol. 17, p. 463, 1979.

Edwards, R. I., "Refining of the Platinum-Group Metals," JOM, p. 4, Aug. 1976.

Edwards, R. I., "Selective Solvent Extractants for the Refining of Platinum Metals," CIM Spec., Vol. 21, p. 24, 1979.

Flett, D. S., "Solvent Extraction in Hydrometallurgy," 1983 Annual AIME Meeting.

Head, M. D. et al., "Nickel Refining by the TBRC Smelting and Pressure Carbonyl Route," 105th AIME Annual Meeting, Las Vegas, NV, 1976.

Hougen, L. R. et al., "Recovery of Nickel, Copper and Precious Metal Concentrate from High Grade Precious Metal Mattes," JOM, p. 6, May, 1975.

Hougen, L. R. et al., "Process for Recovery of Precious Metals from Copper-Containing Materials," U.S. Patent 3,767,760, (1973).

Ireland, D. A., "The First Ten Years of Operations at Impala Platinum Ltd.," Annual CIM Meeting, Vancouver, 1977.

Jolly, J. H., "Platinum Group Metals," USBM Bulletin 671, p. 683, 1980.

Lake, M. J. et al., "Process for Recovering Platinum Group Metals from Ores also containing Nickel, Copper & Iron," U.S. Patent 4,108,639, (1978).

Lawver, J. E., "Mineral Beneficiation Studies and an Economic Evaluation of Minnesota Copper-Nickel Deposit from the Duluth Gabbro," USBM Grant No. G0144109, Mineral Resources Research Center, Minneapolis, 1975.

Montana Department of State Lands and U.S. Forest Service, "Environmental Impact Statement, Anaconda Stillwater Project, Stillwater Co., MT," 1982.

Moore, C. M., "Platinum-Group Metals," USBM Minerals Yearbook, Vol. 1, p. 639, 1980.

Mostert, J. C., "Electric Smelting at Rustenburg Platinum Mines Limited of Nickel-Copper Concentrates Containing Platinum-Group Metals," J. S. African IMM, April, 1973.

Opratko, V. et al., "The Removal of Iron and Arsenic from Leach Liquors at Inco's Copper Refinery, Electrowinning Department," Annual CIM Meeting, Hamilton, 1981.

Plasket, R. P. et al., "Ancillary Smelter Operations and Sulphuric Acid Manufacture at Impala Platinum Limited," J. S. African IMM, Aug. 1976.

Plasket, R. P., "Equipment Selection, Plant Design and Personnel Training at the Impala Nickel-Copper Refinery," Annual CIM Meeting, Vancouver, 1977.

Plasket, R. P. et al., "Recovery of Nickel and Copper from High-Grade Matte at Impala Platinum by the Sherritt Process," Hydrometallurgy, Vol. 3, p. 135, 1978.

Sproule, K. et al., "Treatment of Ni-Cu Matte," Extractive Metallurgy of Cu-Ni-Co, 33, Interscience Publ., London, 1960.

Stern, E. W., "Aqueous Chemistry of Precious Metals Applicable to Refining Processes," Symp. on Recovery, Reclamation & Refining of Precious Metals, Edison, NJ, March, 1981.

Westland, Alan D., "Inorganic Chemistry of the Platinum-Group Elements," CIM Special Volume 23, 1981.

Wong, M. M. et al., "Recovery of Platinum-Group Metals from Stillwater Complex, MT, Flotation Concentrates by Matte Smelting & Leaching," USBM RI 8717, 1982.

Wong, M. M. et al., "Preparation of Platinum-Palladium Flotation Concentrate from Stillwater Complex Ore," USBM RI 8500, 1981.

Chapter VII

POTENTIAL INNOVATIONS IN MINING AND METALLURGY APPLIED TO DOMESTIC RESOURCES OF CHROMIUM, COBALT, NICKEL, MANGANESE, AND PLATINUM, 1990-1992 and 1995-2010

This chapter conceptually and broadly addresses innovation in the mining and mineral processing fields. Reference is made to current technologies more specifically discussed in Chapter V (mining) and Chapter VI (processing). This chapter provides a basis for discussion of future production scenarios discussed in Chapters VIII and IX.

Lead-times in mine preproduction are such that the new mines of 1990-1992 will make use of technology that is already beginning to appear in the 1980's. Innovations in mining and metallurgy that may represent a wider departure from current practice but should still be related in most ways to the new concepts and research of today.

Current-day horizons in mining pertain to improvements in premining investigations, changes in open pit and underground practice, and developments in the new field of in-situ mining.

Innovations for the near future are expected to result in deeper open pits with steep and extensive conveyor systems. Underground mines of the near future can be expected to use an increased number of continuous mining machines for development raise boring. Machinery for continuous operations in shaft sinking, tunnel boring and stoping may be in use, but overall operations will still be dominated by the mining cycle or drill-blast-load-advance. For open pit and underground mining practice available to critical metal resources in the longer 1995-2010 view, continuous mining machinery and rapid conveyor haulage systems may be in wider use than cyclic mining and truck haulage.

In-situ mining has not yet involved the extractoin of chromium, cobalt, nickel, manganese, or platinum. A concept for in-situ leaching of manganese is, however, being developed, and the possibility of in-situ leaching for cobalt, nickel, and chromium is being investigated. The technology of in-situ leaching is becoming well-enough established in uranium and copper to provide a basis for developing similar systems in the critical metal resources. A breakthrough, especially in regard to manganese, is possible, but this would probably not begin to have an impact on domestic resources until the beginning of the longer-term 1995-2010 view.

Metallurgical processing for domestic cobalt deposits could be brought into practice within the near future. There are problems in optimizing cobalt recovery, but the technology is available for sulfide deposits and a technology is being developed for cobalt-nickel-chromium deposits.

Metallurgical processing technology is available for domestic platinum deposits and domestic sulfide nickel bodies; the hurdles in either a near-future view or the longer-term view would be in the economics rather than the technology of extraction. Chromite and manganese metallurgical processes have not yet been developed for low-grade domestic resources on a practical basis; research on these specific commodities is underway, and a breakthrough could provide an impact, especially in the longer term.

HORIZONS IN MINING

Innovations expected to be in general use during the period 1990-1992 will be related to those that are now in evidence as new technology. With a typical lead-time of 5 to 7 years in mine preproduction, today's experimental and pioneering work could become accepted practice in the new mines of the

1990's. Innovations for mining during the period 1995-2010 will probably be related to current new concepts and to research work now underway. The following discussion will first consider new technology now in evidence then turn to longer-term concepts; a specific boundary cannot be drawn between new technology and new concepts.

Innovations can be categorized as follows:

1. Premining investigations--the planning and design work that takes place after an ore body has been sufficiently delineated to provide for a feasibility study and a decision to go into development.

2. Open pit mining.

3. Underground mining.

4. In-situ mining.

Premining Investigations

These investigations, requiring 1 to 3 years of ore reserve definition, geotechnical study, and engineering design, are based on the acquisition and treatment of information. The trend is toward a more accurate and more comprehensive assessment of actual conditions to be met during mining. One effect is to shorten the time needed for full-scale testing and rerouting during development; another effect is to provide a secure basis for estimating capital requirements, operating costs, and financial return. Salient innovations are as follows.

1. The efficiency of ore-reserve drilling and the reliability of the data are being significantly improved by drill hole technology (in-hole

replacement bits, for example) and by the availability of borehole assaying probes (nuclear activation analysis) and borehole geophysical tools.

2. The geostatistical treatment of sample data is now at a point where a large number of configuration and cut-off-grades can be tested and related to alternative mine designs in advance of actual development.

3. Geotechnical testing in drill hole and on-site has become so sophisticated that pit slopes and ground-control methods can be simulated in advance of development work.

4. Computer simulation of an entire mine's productive life can now be used to plan for optimum extraction of ore. Finite element analysis, using block-by-block geotechnical and geologic data, can be related to the mine design.

Open-Pit Mining

Open pit mines dominate in the production of ore, both worldwide and in the U.S. Technology has advanced more distinctly in the last decade than it has in underground mining. The trend is toward large-scale machinery and away from the last of the truck-haulage methods. The cost of energy (especially diesel fuel) and the need for steeper and deeper pits with longer haul roads to the concentrators and dumps has provided another trend—toward more flexible and energy efficient removal of ore and waste from the pit. This latter trend is evidenced by:

1. The use of electric haulage trucks and electric trolley-assisted diesel haul units on the main ramps. The trucks are available in a range of capacities, from 18-tons to 350-tons; shovels with buckets to 30 cubic meter

capacity are available. They are, however, increasingly electric-and-hydraulic operated.

2. High-angle elevating conveyors are replacing truck haulage from some of the major pits (Frizzel and others, 1981). In some instances, a large mobile crushing plant in the pit is used to provide a uniform feed for conveyor belts. Some conveyors have now been designed to take mine-run rock without crushing (Martin and others, 1982). Conveyors are increasingly used for long runs to plants and dumps.

3. Slurry pipelines are beginning to be used for lifting and transporting rock from the pit floor to the plant-site.

Drilling and blasting, once dominated by intensive drill hole patterns and dynamite explosives, have gone to using larger blastholes (to 445 mm), fewer holes, and newer blasting agents. The cycle of drill-blast-load is still the most common mining method, but there is a new consideration for continuous (noncyclic) mining whenever possible. Bucket wheel excavators, already established in coal and softer rock mines, are now being tried in harder rock; with any success at all in hardrock conditions, the bucket wheel excavator--used with belt conveyors--could replace cyclic mining and measurably speed-up production.

Underground Mining

In the machinery of underground mining, the immediate trend is toward improvement in the mining cycle of drilling, blasting, rock support, loading, and hauling. This is manifested by:

1. Larger diameter blastholes and fewer holes per advance.

2. Hydraulic drills and drill jumbo machines, replacing the standard pneumatic machines.

3. More effective blasting with slurries, gels, and ANFO-based explosives rather than dynamite.

4. Rock support in working areas by a new generation of highly effective rock bolts and additional support ahead of mining by grouted bolts (spiles).

5. Remote-controlled loading of rock by trackless load-haul-dump or front-end loader machines, with the operator away from the face.

6. Increased use of conveyor belt haulage, replacing rail and shuttle cars. Newly-available underground crushers are portable; they can now provide a uniform load for conveyor belts and also save in crushing transfers at the surface.

7. Pipeline transport, now in use in a few African, Canadian, and U.S. mines, is being considered for much wider application in conveying and hoisting.

Even though the immediate trend is toward improving the mining cycle, there is a strong trend toward continuous noncyclic mining. This has become established with the use of raise-boring equipment in raises and small shafts. A pilot hole is commonly drilled ahead of a bored raise, but there is now an effective "blind hole" (boxhole) borer.

Continuous shaft-sinking (Davis, 1981) for main hoisting shafts, with impact breakers replacing the drill-blast-hoist sequence, is being done on a pilot scale. If it meets with continued success, this will constitute a major departure in mining technology.

The use of continuous tunnelling machines in hard rock has not yet become economic, but the machines have been successful in soft and medium-hard rock. Hard-rock tunnelling by noncyclic methods should, like continuous shaft sinking, provide an important breakthrough in the speed and cost of mine development.

The ultimate site for continuous mining will be in the stopes themselves. Continuous mining machines have been used in coal and saline beds for a number of years, but they are just now beginning to be tried in hard rock (Sparks, 1980). A "longwall planer", similar to the coal mining machine, has been used at White Pine, Michigan, in copper ore, and a "Dosco" continuous miner is now being installed at the Miami East copper mine in Arizona.

In considering a short time span of 10 to 20 years, cyclic mining will probably continue to dominate underground. In the longer view, continuous mining should be much more widely used; it will certainly not be applicable in all sites, but it is a developing trend (Howard, 1975; Kostuik, 1980).

Underground mining methods and configurations will probably continue in the established pattern, with the available approaches distributed between the more costly methods of sublevel caving shrinkage, and cut-and-fill and the lower-cost methods of block caving, open stoping, room-and-pillar, and sublevel stoping (Dravo Corp., 1974). Two methods newly in use should, however, have an impact on future underground mining, these are Vertical Crater Retreat and Ramp-in-Stope.

The vertical crater retreat system (Lang, 1982) now in use in several Canadian mines and in the Homestake Mine in the U.S., is revolutionary in that it makes use of sperical (short-length) charges blasted in long drillholes from a level above the production level. The charges are fired in a series of holes, and the blasting is repeated as the lower-level stope progresses upward. In many respects, vertical crater retreat is a major improvement in the sublevel stoping pattern. There is a minimum of development work between levels, fragmentation is thorough, and there is little dilution with barren wall rock. At the Homestake gold mine, productivity is double that of cut-and-fill mining and nearly double that of conventional sublevel stoping.

The ramp-in-stope method (Rasmussen and Pugh, 1979) is a mechanized cut-and-fill approach that is especially well-suited to narrow veins where some support of the wall rock is needed. In this approach, still in the testing stage, trackless equipment can be moved quickly along an inclined surface in fill material while stoping continues rapidly upward in a narrow vein.

In-Situ Mining

In-situ mining constitutes the solution or leaching in place of ore minerals and the erosion of material from the bottom of a drill hole. The first approach, in-situ leaching, is related to the established practice of solution mining in saline deposits; it is now a fast-developing method of copper and uranium recovery. The second approach, borehole mining, is still in an experimental stage. Both approaches involve the recovery of ore through a series of drill holes rather than from active workings at depth.

There are approximately 30 active uranium in-situ leaching projects in the United States and there are a number of in-situ copper recovery operations. The leaching of copper minerals and of gold in dumps or heaps on the surface is a closely allied procedure, and many of the basic chemical principles are similar.

In-situ leaching (Aplan and others, 1974; Bhappu, 1976; and Davidson, 1980) is especially well-suited to small or low-grade ore bodies because of its relatively low capital requirement and short development time. The ore body must either have an inherent permeability or must be capable of being made permeable by blasting or hydraulic fracturing. Blasting and hydraulic fracturing from drill holes are most effective within several hundred meters of the surface, where the broken rock can expand or swell. Permeable conditions may be found in the wall-rock of abandoned mine workings; this is a current situation in copper in-situ leaching.

Permeable conditions may also be induced by the caving or shattering of rock in a preparatory underground mining operation; this is still a conceptual situation. The use of nuclear explosion in ground preparation for in-situ leaching has been considered. One such project has been planned in a large copper ore body in Arizona, but the project has been suspended.

The leaching solutions, acidic or alkaline solvents, are circulated within a permeable bed or zone by a geologic condition of tighter surrounding rock, by hydrologic configuration in the injecting and producing wells, and by grout curtains between peripheral drill holes. The U.S. Bureau of Mines is now experimenting with various well configurations and pumping rates in a series of geologic conditions.

Recovery of metal from solutions at producing wells can be made by cementation (existing copper in-situ operations) or by solvent extraction (existing uranium in-situ practice). In-situ leaching has been successful in copper and uranium mining. It is being proposed for the mining of deep channel-type gold deposits in Australia, and it is now being investigated for applications to nickel, manganese, cobalt, molybdenum, chrome, and silver deposits (Agarwal and Burrows, 1981; Anon, 1982-Mining Engineering).

Borehole mining, or slurry mining (Dravo Corp., 1974; Davis, 1981) makes use of a water-jetting and slurry-pumping system. As developed by the U.S. Bureau of Mines and also as used in the "Marconajet" miner, the water jet erodes the ore to form a slurry that is pumped to the surface. The system has been tested in friable uranium-sandstone deposits and in tin-weathered granite deposits. From a single well unit, an operating radius of 12 m has been obtained, with a production of 75 tons per hour. A full-scale production installation would comprise an entire field of wells. Borehole mining should be applicable to shallow deposits in relatively friable rock.

MINING APPLICATION TO CHROMIUM, COBALT, NICKEL, MANGANESE, AND PLATINUM DEPOSITS

Chromite Deposits

Chromite deposits in the United States are in lateritic material (Gasquet Mountain), in steeply dipping hard-rock veins (Stillwater Complex), in small podiform hard rock bodies, and in beach sands. In the hard rock ore bodies, open pit and underground mining systems would be similar to those used in other ore bodies; the trends to increased mechanization and to continuous mining systems would apply. The vertical crater retreat system would be especially applicable in narrow and steeply dipping veins and podiform bodies. In the podiform chromite deposit of Greece (Sassos, 1982), the wall rock is competent, allowing for sublevel open stoping--and potentially a vertical crater retreat system.

In lateritic material and bench placer type sands, open pitting would be very likely to involve continuous mining by bucket wheel machines or by shovels without the need for drilling and blasting. At the Riddle nickel deposit in Oregon, shovel mining of laterite did not require blasting.

Solution mining of chromite is still in the conceptual stage, but if experimental work is favorable, this could provide an approach to the mining of hard rock chromite with explosive fracturing or to the mining of lateritic deposits that are inaccessible for open pitting.

Cobalt Deposits

Cobalt deposits are in lateritic rock (Gasquet Mountain-cobalt byproduct), in steeply dipping hard-rock bodies (Blackbird), in bedded hard rock bodies (Missouri), and in the gently-dipping hard rock of the Duluth Complex. Lateritic deposits are suited to open pit mining and to the continuous systems of excavators and conveyor belts that will become more common in steep pits. Open pit mining in harder rock would be practicable in the copper-nickel-cobalt deposits of the Duluth Complex, with underground mining at depth. Underground mining in the competent rock of the Duluth Complex would involve open stoping and room-and-pillar methods. Steeper hard rock bodies of cobalt ore would be in competent rock or, as at Blackbird, in heavily fractured ground requiring support. In the latter condition, cut-and-fill mining and the new ramp-in-stope system could be appropriate.

Nickel Deposits

Nickel deposits in the Duluth Complex (Kreismann, 1980) in the laterites, and in Missouri would be mined in the same way as metioned for cobalt.

Solution mining of nickel and cobalt is still an untested concept, but it could provide an important breakthrough.

Manganese Deposits

Manganese deposits in the U.S. are in a variety of potential mining evironments, ranging from relatively soft and incompetent lake bed deposits to steeply-dipping bodies in hard rock. Mining would be by the standard and new methods for ore deposits in general. Solution mining of manganese ore, while still in the conceptual stage, is being investigated directly by the U.S. Bureau of Mines. In the Bureau of Mines approach, manganese ore in the Artillery Peak area, Arizona, would be undercut and blasted to induce permeability the permeable block would then be leached with sulfur dioxide. Manganese would be extracted from solution as a sulfate or oxide.

Platinum Deposits

Platinum deposits of potential economic grade are in steeply dipping and thin hard-rock zones (Stillwater Complex), irregular hard rock masses with disseminated platinum group minerals (Alaska), and in placers (Alaska). Mining of the hard rock deposits would be by underground methods, with sublevel stoping and vertical crater retreat systems to be considered.

HORIZONS IN METALLURGY

As with mining, metallurgy innovations during the next decade will undoubtedly be related to those that are now in evidence as new technology. Innovations during the longer time frame, 1995-2010, will be more related to concepts now being formulated and research work now underway. A new metallurgical plant can ordinarily be brought on-stream in less time than the

5 to 7 years needed for the development of a large mine; the two are, however, so closely related that the pre-production time is effectively the same.

The following discussions will deal with new trends in technology and with new concepts; as with horizons in mining, a precise boundary cannot be drawn between the two.

Innovations in metallurgy can be categorized as dealing with three stages:

1. Mineral processing--the treatment, beneficiation, or milling of ore to provide a higher-grade mineral concentrate for subsequent treatment.

2. Extraction--the reduction of minerals or mineral concentrates to metals by pyrometallurgical (smelting) or hydrometallurgical (chemical) methods.

3. Refining--the production of marketable metals from the products of extractive metallurgy.

It is recognized that not all ores will require all three stages of treatment. Some run-of-mine ores, nickel laterites and chromite for example, are commonly smelted directly. Some metals are taken directly into solution from the ore and recovered from the solution as refined metal.

Mineral Processing

The principal steps are crushing, grinding, and separation. In all of these, there is an increasing emphasis on the effective use of energy.

The effectiveness of crushing units using mechanical energy directly on the ore has not changed substantially, but there are significant concepts in crushing technology that make use of nonmechanical energy. These new methods, still in the experimental stage, make use of thermal and hydraulic shock, high-voltage electrical discharge, and high energy particle bombardment. At one mine and mill in the Soviet Union, high voltage shock is now used to break hard ore and lessen the burden in crushing (Mollick, 1980).

In the grinding of crushed ore, there is a specific trend toward wider use of autogenous and semiautogenous units that use the ore to grind itself in rotating mills. These units have been available for a number of years, but there is now a stronger emphasis on replacing ball .pamills and rod mills with more economical autogenous grinding whenever the type of ore will permit.

Separation of the ore minerals from gangue is accomplished by gravity, electromagnetic, electrostatic, and flotation methods, with flotation as the most widely used method. Flotation machines are now larger in size and fewer in number for a given amount of ore, and flotation reagents are increasingly effective. Flotation and gravity separation are now being used in combination for ores that have several co-products; in one such instance, a chromite concentrate is recovered by gravity separation and a platinum-gold concentrate is recovered from the tailing by flotation (Martinez and others, 1981).

In preparing a concentrate for subsequent extraction metallurgy, calcining and agglomerating or nodulizing systems are coming into increased use. In one such system, a rotary kiln is used to calcine a manganese carbonate concentrate to manganese oxide pellets for furnacing. Laterite ores are commonly dried and calcined in kilns before smelting (Berry and Turner, 1982).

Extraction and Refining

Pyrometallurgical extraction is the long-established method, and it is applied to sulfide or arsenide and oxide ores and concentrates. The feed material, which may be roasted to oxidize iron and drive off sulfur dioxide, is smelted in an electric, reverberatory or flash furnace to produce an impure metallic matte; the matte is then melted with silica and blown with air in a converter to remove most of the remaining iron as slag and to remove most of the sulfur as sulfur dioxide. The converter matte is refined by electrolytic or hydrometallurgical methods. In some situations, as with a nickel-copper converter matte, the matte is crushed and the components separated by flotation before continuing with the refining process.

Innovations in pyrometallurgy will involve the increased use of oxygen in flash smelting and, as with mining and mineral processing, will be directed toward continuous rather than cyclic operations.

Hydrometallurgical extraction involves alkaline or acid leaching of sulfide concentrates and oxide ores, with the metals being recovered from leach liquors by chemical precipitation, ion exchange, or electrowinning. Innovations in hydrometallurgy will apparently involve the upgrading and purifying of leach liquors, making them increasingly suited to the recovery of pure oxides or salts for thermal reduction into metal and fabricated shapes.

METALLURGY APPLICATIONS TO CHROMIUM, COBALT, NICKEL, MANGANESE, AND PLATINUM DEPOSITS

Chromite

Chromite for metallurgical use should contain at least 45% Cr_2O_3 and have a Cr:Fe ratio of at least 2.5:1. This is obtained in foreign ores and gravity concentrates but has not ordinarily been obtained from the processing of domestic chromite deposits.

Low-cost methods of beneficiating domestic deposits to an acceptable concentrate have been studied for a number of years by the U.S. Bureau of Mines. The methods have involved combinations of gravity and electrostatic separation plus flotation to obtain a higher chromium content, with leaching to reduce the iron content. Research has not yet provided for an economic method of up-grading. A direct smelting process for Stillwater Complex ore has been investigated; this .pawould provide a high-iron alloy, but still not comparable in grade and cost with imported ferrochrome (U.S. Bureau of Mines, 1980).

Lateritic ores containing chromium are ordinarily roasted and leached. An experimental plan for the recovery of chromium from laterite ores, as at Gasquet Mountain, involves roasting and leaching after gravity beneficiation, with final electrowinning for nickel and cobalt and final recovery of chromium from the leach residue. Future research and experimentation in chromite recovery and chromium extraction will most probably involve hydrometallurgical processing (U.S. Bureau of Mines, 1980).

Cobalt

Cobalt in sulfide, arsenide, or oxide mineral form is processed in a variety of methods involving a combination of magnetic concentration, flotation, roasting, leaching, and chemical or electrostatic precipitation. Foreign ore bodies contain 0.1% to 0.5% cobalt, generally as a recoverable byproduct in base metal or precious metal ores. Ore from the Duluth Complex, also associated with copper and nickel sulfide, has an even lower cobalt content. Cobalt deposits in Missouri, also associated with base metal sulfides, have a low cobalt grade, but would probably be processed in a similar manner to the Duluth and foreign ores. The cobalt grade at the Blackbird Mine, Idaho, is relatively high, approximately 0.6%, but the recovery of a high-grade cobalt concentrate from the Blackbird copper-cobalt-iron sulfide ore is reported to be difficult (U.S. Bureau of Mines, 1981).

A method is being investigated for the recovery of a low cobalt content from the Gasquet Mountain laterite. Current research on hydrometallurgical processes for the recovery of cobalt from manganese nodules on the ocean floor may be transferrable to lateritic deposits on land.

Nickel

Nickel in sulfide ore is generally amenable to concentration by established flotation methods, but there is not, as yet, an acceptable means of concentrating lateritic nickel ore. Nickel sulfide concentrates treated by hydrometallurgy are leached with ammonia; copper, nickel, and cobalt are precipitated by heating and hydrogen treatment. Lateritic nickel ores are ordinarily smelted directly, leached with ammonia or leached with sulfuric acid.

Nickel ore in the Duluth Complex would probably be treated in a similar way to foreign sulfide ores. Nickel from domestic laterite deposits would be treated by gravity beneficiation, roasting, and leaching.

Manganese

Nearly all manganese ore undergoes some form of beneficiation. Washing, screening, gravity separation, magnetic separation, and flotation have been used. Carbonate manganese ores require calcining and nodulizing or pelletizing.

Chemical and roasting processes have been developed for beneficiating the low grade manganese resources in the United States, but these have so far been too costly for extended use. Manganese nodules of the ocean floors have been a subject of metallurgical research but the manganese content does not appear to be as economically attractive as the nickel, cobalt, and copper content. Research in the potential in-situ leaching of manganese deposits indicates that recovery could possibly be made using an aqueous solution of sulfur dioxide as a leaching agent (U.S. Bureau of Mines, 1980).

Platinum

The treatment of South African platinum ore, with nickel and copper as associated products, involves flotation and smelting to a nickel-copper matte with a high platinum value. The matte is treated electrolytically, with platinum collecting in the anode slimes or residues after recovering copper and nickel. The resulting platinum concentrate is leached with aquaregia. A South African method for treating a platinum flotation concentrate from chromite-rich ore involves high temperature smelting and chemical extraction (U.S. Bureau of Mines, 1980); this method is being investigated in relation to the platinum resources of the Stillwater Complex.

CONCLUSIONS

Innovations in mining and metallurgy available for domestic chromite, cobalt, nickel, manganese, and platinum resources during the period 1990-1992 will be directly related to current trends in technology. Lead-times in mine preproduction are such that the mines of the 1990's can only make use of practices that are beginning to appear in the 1980's. Innovations for mining and metallurgical practice in the period 1995-2010 will probably be related in most ways to the new concepts of today and to the research that is now underway. Mining, as the heaviest of the heavy industries, requires decades of lead-time for concept to research, research to pilot operation, and pilot operation to general practice.

1990-1992

The overall picture for mining technology available to domestic chromite, cobalt, nickel, manganese, and platinum resources should differ from that of today only in a few respects. Open pit mining will be using more continuous conveying systems in deeper and steeper pits. In some pits, continuous bucket-wheel excavators will have come into use, but the major work will continue to be done by drilling, blasting, and loading rock. Underground mining should still be a basically cyclic operation of drilling, blasting, and loading, but there will be increased remote control, rapid conveyor haulage, and mining methods that break rock on remote levels. Raise boring machines should be in general use, but the remaining continuous mining innovations in shaft sinking, tunnel boring and stope production will probably be used in a few mines.

In-situ leaching may have begun in manganese deposits but only in those deposits that are ideally suited to the method. In-situ leaching of nickel, chromium, and cobalt could be in a pilot operation stage.

Metallurgical processing of domestic cobalt deposits could be brought underway with the processes not too different from those now available. If the need for nickel, chromium, and manganese production were to become acute, there could still be major problems in economic extraction.

1995-2010

The picture for technology available to the extraction of critical metal resources will have become that of continuous processes in open pit mining, underground mining, and metallurgical plants. Open pit mining may have advanced to a wide use of excavators and conveyor or pipeline transfer of rock; many pits will, however, probably require continued use of drilling and blasting. Underground mining may have gone to a dominant use of continuous mining machines, and haulage systems may have gone entirely to rapid conveyors and pipeline transfer.

In-situ leaching technology may have developed to a point where manganese, nickel, and cobalt could be extracted on a limited scale. Metallurgical processing could be in a position to accommodate the critical metals, but perhaps on a limited basis and with high cost.

No provision has been made for a totally unexpected breakthrough in technology. If such were to occur, it would still take a decade or more to have a broad impact on mining practice.

Economic Considerations

Changes in technology should affect the cost of chromium, cobalt, nickel, manganese, and platinum production by reducing the level of preproduction investment and by reducing the unit cost of operating. The anticipated technologic changes are, however, capable of being used on a worldwide basis.

Domestic resources of the critical metals are not so geologically unique that the same innovations would not apply to foreign deposits of higher grade. Thus, in a business-as-usual scenario for the availability of critical metals from foreign sources, new technology would not be expected to make the metals from domestic sources more competitive. Without discoveries of higher-grade ore bodies in the United States, marginally economic deposits would remain as such.

In a situation with foreign supplies restricted or unavailable, the new technology would provide for domestic production at lower costs than would be experienced using current technology. In comparison to future capital and development costs without significant technologic change, the preproduction investment in open pit and underground mines could be reduced by as much as 15% in the 1995-2010 period with the use of rapid excavation and continuous material-handling methods. Judging from existing uranium and copper projects, the preproduction investment for in-situ leaching would be considerably lessened, perhaps by 50%, in comparison with conventional open pit and underground mine installations using present-day technology. Operating costs per unit of material in open pit and underground mines should be lessened by as much as 20% in relation to the costs that would apply using current technology. In-situ leaching operating costs should be markedly reduced, perhaps to 40% below that of conventional mining and milling with present-day technology.

In essence, innovations in mining and metallurgical technology are not expected to make domestic production of the critical metals from existing sources more competitive with imported material, but the innovations can be of considerable economic importance in the event of a cut-off in supply.

BIBLIOGRAPHY

Agarwal, J. C. and Burrows, J. C., 1981, Economic impact of in-situ solution mining of copper, paper 4, in National and International Management of Mineral Resources: London, Inst. of Min. and Metall., p. 1-7.

Aplan, F. F. et al., 1974, Solution Mining Symposium: New York, AIME, 469 p.

Anon, 1982, Solution mining: Mining Engineering, Vol. 34, No. 5, p. 516-518.

Berry, W. W. and Turner, A. L., 1982, New horizons in mineral processing: Mining Congress Journal, Vol. 68, No. 6, p. 56-60.

Bhappu, R. B., 1976, In-situ mining--theoretical and practical aspects, in Recent Advances in Mining and Processing of Low Grade and Submarginal Mineral Deposits: New York, Pergamon Press, p. 24-46.

Davidson, D. H., 1980, In-situ leaching of non-ferrous metals: Mining Congress Journal, Vol. 66, No. 7, p. 52-57.

Davis, L.L., 1981, Bureau of Mines research to improve underground metal/non-metal mining technology: Mining Engineering, Vol. 33, No. 3., p. 305-312.

Dravo Corp., 1974, Analysis of Large Scale Non-Coal Underground Mining Methods: U.S. Bur. Mines Open File Report 36-74, 605 p.

Frizzell, E. M. et al., 1981, Trucks versus high angle conveyor haulage in open pit mines: Mining Congress Journal, Vol. 67, No. 10, p. 19-26.

Howard, T. E., 1975, Underground mining and rapid excavation: Mining Engineering, Vol. 27, No. 6, p. 37-42.

Kostuik, J., 1980, Mining in the next decade: Canadian Mining and Metallurgical Bulletin, Vol. 73, No. 823, p. 48-54.

Kriesmann, P., 1980, Introduction to Minnesota's Copper-Nickel study: AIME Preprint 80-323, 9 p.

Lang, L. C., 1982, Vertical crater retreat--an important new mining method, in Hustrulid, W. A., ed., Underground Mining Methods Handbook: New York, AIME.

Martin, T. W. et al., 1982, Large rock conveying stysems and their application in open-pit mines: Mining Engineering, Vol. 34, No. 10, p. 1481-1485.

Martinez, E. et al., 1981, Concentration: Mining Engineering, Vol. 33, No. 5, p. 529-531.

Mollick, L., 1980, Crushing: Engineering and Mining Journal, June, p. 96-103.

Rasmussen, D. G. and Pugh, G. M., 1979, Ramp-in-stope access: Engineering and Mining Journal, August, p. 79-84.

Sassos, M. P., 1982, Greek chromite--Hellenic ferroalloys: Engineering and Mining Journal, October, p. 84-87.

Sparks, G. B., 1980, Application of hard rock continuous miners to cut-and-fill slot stoping: Mining Congress Journal, Vol. 66, No. 5, p. 29-33.

U.S. Bur. of Mines, 1980, Mineral Facts and Problems: USBM Bull. 671.U.S. Bur. of Mines, 1981, Minerals Yearbook.

Chapter VIII

U.S. METALS PRODUCTION IN THE YEAR 1995

CHROMITE ANALYSIS

U.S. Bureau of Mines-supplied data (1982) presented in the following four tables (Tables 8-1, 8-2, 8-3, and 8-4) demonstrate the capacity of U.S. resources to produce chromite ores in the 1995 time frame. Further analysis can be found in the recently published USBM Information Circular 8895, entitled Domestic Chromite Availability (1982). The resource base in the United States can be defined as stratiform and stratiform-disseminated deposits, podiform and podiform-disseminated deposits, chromite-laterite deposits (together with nickel and perhaps cobalt), and beach placers. The stratiform and disseminated-stratiform deposits are essentially exclusively the chromite ores of the Stillwater Complex of Montana. Podiform and disseminated-podiform ores are located primarily in the northern California and southern Oregon, as well as Alaska. The lateritic chromite ores are primarily the result of weathering of these podiform deposits, as wll as ultrabasic rocks containing small chromite concentrations. Placer beach and stream sands are located in Oregon and Maryland/Pennsylvania.

Currently available resource data suggest that the ores of Gish and Mouat/Benbow mines in the Stillwater Complex probably contain a grade of ore between 10% to 20% CR_2O_3, with an average grade of perhaps 15%. The Mouat/Benbow deposit contains the much larger of the two known resources and is in the range of 25 million tons combined demonstrated and inferred resources. USBM projected mine life planning for the Mouat/Benbow deposit is an estimated 46 years at recovery rates of perhaps 300,000 tons per year of ore, and suggests 1.5 million tons of total recoverable CR_2O_3 at a 70%

TABLE 8-1
CHROMIUM INFORMATION

State Deposit	Type[1] of Deposit	Status[2]	Owner/Operator	Past Production	Proposed Mining Method	Proposed Processing Method	No. of Pre-Prod Years	Proposed Mine Life Years
Alaska								
Claim Point	Pod/Diss	Ppd	Union Carbide	NA	Open Pit	Gravity	4	14
Red Bluff Bay	Pod/Diss	Exp	NA	None	Open Pit	Gravity	2	4
Red Mountain	Pod/Diss	Ppd	R.S.Richards/Union Carbide/ Kenai Chrome Co.	1943-1957	Overhand Shrinkage	Gravity	2	11
California								
Bar Rick Mine	Pod/Diss	Ppd	Collins H. McClendon/ Inspiration Devel. Co.	1953-1958	Sublevel Stoping	Gravity	2	13
Pilliken Mine	Pod/Diss	Ppd	Red Line Transfer Co.	1915-1957	Open Pit	Gravity		
McGuffy Creek	Pod/Diss	Ppd	Elmer Weeks	1941-1958	Open Pit	Flotation	2	4
Seaid Creek/ Emma Bell	Pod/Diss	Ppd	U.S. Chrome	1918-1956	Strip Hillside	Gravity	3	9
North Elder Creek Area	Pod/Diss	Ppd	California State	1886-1954	Open Pit	Gravity		
Georgia								
Louise Chromite	Placer[1]	Exp	Numerous Miscellaneous	None	Open Pit	Gravity	1	6
Maryland								
West Placer Area	Placer[1]	Ppd	Mrs. Paul West	NA	Open Stope	Magnetic/ Electrostatic	1	15
Montana								
Gish	Strat/Diss	Ppd	Monte Vista Co.	NA	Open Stope	Flotation	3	3
Mouat/Benbow	Strat/Diss	Ppd	Anaconda	NA	Open Stope	Flotation	3	46
North Carolina								
North Carolina Area	Placer	Ppd	NA	NA-1918	Open Pit	Gravity	1	5

TABLE 8-1,
CHROMIUM INFORMATION
(continued)

State Deposit	Type[1] of Deposit	Status[2]	Owner/Operator	Past Production	Proposed Mining Method	Proposed Processing Method	No. of Pre-Prod Years	Proposed Mine Life Years
Oregon S.W. Oregon Beach Sands	Placer	Ppd	NA	1943-1943	Strip Hillside	Magnetic/ Electrostatic	2	11
Pennsylvania Renshaw Placer	Placer	Ppd	NA	1943-1975	Open Pit	Gravity	1	5
Wyoming Casper Mountain	Strat/Diss	Exp	Consolidated Mining Co./ Wyoming/Baptist Conv./ City of Casper	None	Bench (BERM)	Gravity	3	11

NA--Not Available

[1] Pod--Podiform, Strat--Stratiform, Diss--Disseminated

[2] Ppd=Past Producer, Exp--Explored Deposit, Dev--Developing Mine

#0082A
Source: USBM MAS File, 1982.

TABLE 8-2 - CHROMITE RESOURCES
(thousand metric tons)

State	Property	Grade Cr_2O_3 percent	Demonstrated Mineralized material	Demonstrated Contained Cr_2O_3	Demonstrated Recoverable Cr_2O_3	Inferred Mineralized material	Inferred Contained Cr_2O_3	Inferred Recoverable Cr_2O_3
AK	Claim Point............	17.8	267	47.6	31.3	--	--	--
	Red Bluff Bay..........	12.0	30	3.6	2.9	--	--	--
	Red Mountain...........	25.8	--	--	--	183	47.2	34.0
CA	Bar Rick Mine..........	7.6	5,065	384.9	294.5	39,447	2,998	2,293.5
	McGuffy Creek[1].......	W	W	W	W	W	W	W
	North Elder Creek[2].....	11.9	--	--	--	104	12.4	9.5
	Pilliken Mine..........	5.0	--	--	--	30,975	1,548.8	1,184.8
	Seiad Creek/Emma Bell..	5.0	4,546	227.3	173.9	6,280	314.0	240.2
GA	Louise Chromite........	0.4	131	0.6	0.5	--	--	--
MD/PA	West Placer Area[3]......	1.4	729	10.1	7.7	--	--	--
MT	Stillwater Complex							
	Benbow/Mouat.........	W	W	W	W	W	W	W
	Gish Mine............	15.0	500	75.0	57.4	354	53.1	40.6
NC	North Carolina Area[4]...	1.9	108	2.1	1.6	70	1.40	1.1
OR	Southwest Oregon Beach							
	Sands................	5.6	10,827	604.1	462.2	34,945	1,950.0	1,491.8
PA	Renshaw Chromite.......	1.7	209	3.5	2.7	--	--	--
WY	Casper Mountain........	2.5	3,774	92.5	67.4	--	--	--
	Total[5]...............	NAp	46,604	5,620.6	4,257.5	194,019	19,333.2	5,295.5

(NAp - not applicable; W - withheld to avoid disclosing individual company proprietary data; included in total).

[1] Domestic - chromite reserve base.
[2] Includes 3 deposits which have been combined for analysis.
[3] Includes 13 deposits which have been combined for analysis.
[4] Includes 3 deposits which have been combined for analysis.
[5] Includes resources withheld to avoid disclosing individual company proprietary data.
Source: USBM Mass File, 1982.

TABLE 8-3 - CHROMITE-LATERITE-DEPOSIT RESOURCES[1]
(thousand metric tons)

State	Property	Grade Cr_2O_3 percent	Demonstrated			Inferred		
			Mineralized material	Contained Cr_2O_3	Recoverable Cr_2O_3	Mineralized material	Contained Cr_2O_3	Recoverable Cr_2O_3
	Pine Flat Area							
OR	Eight Dollar Mountain...	1.1	--	--	--	13,023	145.9	77.3
CA	Gasquet Laterite........	W	W	W	W	W	W	W
CA	Little Rattlesnake......	W	W	W	W	W	W	W
CA	Lower Elk Camp..........	W	W	W	W	W	W	W
CA	Pine Flat Mountain......	2.8	6,382	178.7	94.7	8,670	242.8	128.7
OR	Red Flat................	W	W	W	W	W	W	W
CA	Red Mountain............	W	W	W	W	W	W	W
OR	Rough and Ready.........	1.5	--	--	—	5,931	90.7	48.1
OR	Woodcock................	1.3	--	--	--	8,587	112.5	59.6
NAp	Total[2].................	NAp	33,813	640.0	602.50	143,126	2,995.4	2,387.6

(NAp - not applicable; W - withheld to avoid disclosing individual company proprietary data; included in total).

[1] Not included in cost-price analysis because laterite-process technology has not been commercially proven.
[2] Includes resources withheld to avoid disclosing individual company proprietary data.

Source: USBM MAS File, 1982.

TABLE 8-4
CHROMIUM COSTS
(January 1982 Costs)

Type & Size of Operation	No of Deposits	OPERATING COSTS Average Mine	Range	Average Mill	Range	$/MT ORE Average Total	Range	CAPITAL COSTS MILLIONS $ Average Total	Range	AVERAGE ANNUAL PRODUCTION MT. Ore	MT. Recoverable Chromium
Placer Operation 25,000 - 1,000,000 MTPY	4	3.00	1.80-4.00	8.10	5.60-11.70	11.30	8.40-15.30	5	1-23	281,000	10,773
Small Open Pit Op 9,000 - 25,000 MTPY	3	33.10	3.60-50.80	20.20	3.60-28.50	52.60	9.5-77.10	4	1-12	17,000	1,168
Large Open Pit Op 150,000 - 565,000 MTPY	3	10.00	7.40-12.80	4.80	4.40-5.50	14.70	12.90-17.20	22	15-29	364,300	12,827
Underground Op Over 175,000 MTPY	3	31.70	18.60-38.80	6.70	5.70-8.20	38.40	24.80-47.00	24	13-39	350,000	48,473

Source: USBM MAS File, 1982.

recovery rate. This is perhaps the largest single U.S. resource of chromium. Using USBM estimates, the average operating cost per metric ton of these ores at about $25.00 in 1982 dollars, with a capital investment cost of about $25 million and a typical capital cost per annual metric ton of about $37.00 (table 8-4). Potential annual production of 300,000 tons of chromite ore containing 185,000 metric tons of chromite per year would amount to about 3% of U.S. chromite demand (Table 8-5).

The underground mining operations of the Stillwater Complex would produce the lowest cost chromium in the United States when one combines both operating and capital costs over operating mine life (table 8-6). The potential for mine production of chromium on an annual basis would be the most that could be recovered from varying individual chromium resources within the United States.

Another U.S. chromium type occurrence is in the podiform or podiform/disseminated chromite ores of northern California. Most of these ores would be amenable to open pit mining (i.e. Pilliken Mine) and involve average capital costs ranging from as high as $20 million per mine for large open pits to as low as $4 to $5 million for a small pit operation. Average annual operating costs for ores of this type may be as low as $15.00 per metric ton for large open pits, with a range between $13.00 to $15.00, to as high as $52.00 per metric ton (underground mines) with a range from $10.00 to $80.00. In addition, the mine-life per ton costs (operating and capital) of chromium for podiform deposits appears to be 50% higher than for the Stillwater ores (table 8-6). The operation of a number of the podiform chromite deposits could produce as much ore annually as the stratiform deposit in the Stillwater Complex. However, ore-grades averaging 5% to 6% CR_2O_3, are considerably lower than the Stillwater, and the total amount of recoverable

TABLE 8-5
UNITED STATES NET IMPORT POSITION FOR CHROMITE AND FERROCHROMIUM
(000 tons gross weight)

	Chromite					Ferrochrome				
	Imports	-	Exports	=	Net Imports	Imports	-	Exports	=	Net Imports
1971	1,299		180		1,119	85		9		76
1972	1,056		77		979	150		13		137
1973	931		55		876	168		15		153
1974	1,102		117		985	170		8		162
1975	1,252		184		1,068	323		13		310
1976	1,275		209		1,066	259		14		245
1977	1,336		248		1,088	228		12		216
1978	1,013		52		961	322		19		303
1979 e	1,150		26		1,124	200		12		188

Source: Dames & Moore, 1980, p. 40. Data based on USBM Mineral Commodity Summaries, 1975-1980.

e = estimate

Table 8-6
Chromium Cost Analysis by Deposit Type

Operating Cost

Deposit Type	$MT ore	CR grade	$MT Cr	$MT Cr_2O_3
Placer	$11.30	.038	$297	$202
Small Open Pit	$52.60	.069	$762	$518
Large Open Pit	$14.70	.035	$420	$286
Underground	$38.40	.138	$278	$189

Capital Cost

$/Annual Cr Prod MT	Cr Inc. Cost & Payback	Assumed Mine Life	Cr Mine Life	Cr_2O_3 Mine Life
$4641	$928	8 yrs	$116	$79
$3465	$693	4 yrs	$173	$118
$1715	$343	11 yrs	$31	$21
$495	$99	18 yrs	$5.5	$3.70

TOTAL COSTS

	Operating Cost	Capital Cost	Total	
Placer	$202	$79	$281	1.5
Small Open Pit	$518	$118	$636	3.3
Large Open Pit	$286	$21	$307	1.6
Underground	$189	$4	$193	1.0

1978	125/$MTCr_2O_3$
1979	88
1980	81

chromite produced would be less than that recoverable from the Stillwater deposit.

Chromite resources in podiform deposits are rather small, the exception being in the Bar Rick mine which could perhaps produce for a dozen or more years on a demonstrated and inferred resource base estimated to be around 3 million tons of 7% CR_2O_3 (see Table 8-2). The second largest of these deposits, one that operated intermittently over a period from 1950-57 is the Pilliken mine near Sacramento, California. This particular deposit has an estimated containd CR_2O_3 resource base of 1.5 million tons, but the ore grade is reported to average only 5% CR_2O_3.

No other known podiform deposit can approach either the contained tonnage or the low mining cost of the Bar Rick and Pilliken properties, and they appear to be the only potentially minable ores of this type in the time frame of interest. However, these chromite ores can probably be produced at costs significantly less than those of the Stillwater Complex. In addition, when considering shipping costs of U.S. western chromite ores (between \$10.00 and \$50.00/ton by rail to the East coast); it is quite clear that the international chromite market can double the delivered price of chromite and still compete with ores derived from the best of the deposits in the U.S., namely the Stillwater, Bar Rick and Pilliken mines of northern California. The building of a ferrochrome smelter in northern California, however, could somewhat alter the economics of a chromite mining industry. There is little likelihood for such construction in the next decade, however, given the high cost of local ores and their noncompetitive position.

The U.S. also has a chromium resource base in the chromitiferous laterite deposits of northern California and Oregon. It is clear that potentially recoverable chromite in these deposits is one to two orders of magnitude less than those of the podiform and stratiform deposits discussed above. Ore-grades are extremely low, generally ranging from 1% to 3% CR_2O_3, and are expected to be extremely erratic in individual deposits. Also, appropriate recovery technology has not yet been fully developed for these chromite ores, nor economically demonstrated, although gravity and flotation techniques will most likely be successful. These ores will not be recovered as a primary chromite resource, but rather as a byproduct in the mining of the nickel/cobalt rich laterites that also contain chromite; such a plan has been developed at Gasquet Mountain. The amount of chromite that could be recovered from laterite deposits in the U.S. would be a very small fraction of demand, and it would not affect our foreign dependency for chromite supply. It is highly unlikely, therefore, that one could mine these small resources economically for primary chromite at costs comparable to a podiform or stratiform deposit.

The Oregon beach sands contain a rather large amount of demonstrated and inferred chromite, estimated to be in the neighborhood of 3 million tons of contained CR_2O_3. Although this resource base is comparable in size to the other U.S. deposits, ore-grades are generally very low, commonly less than 1% CR_2O_3, but in places might run as high as 5% CR_2O_3. The onshore and nearshore Oregon beach sands present enormous institutional and environmental problems associated with their mining. A small amount of chromite was produced, however, from this resource during World War II.

Placer operations of the type that could be undertaken to recover chromite ores of the Oregon beach sands would have an average operating cost of around $11.00 per metric ton of ore, and a rather small average capital investment of about $5 million in 1982 dollars. Annual recoverable tonnage could be comparable to that from two or three of the largest podiform deposits of northern California, and with somewhat lower ore grades would provide essentially equivalent amounts of recoverable CR_2O_3, estimated to be about 10,000 to 50,000 tons per year of the CR_2O_3. Such production would provide a relatively small part of U.S. chromite demands (1%) and expected mine life is measured in the order of 10 to 15 years (see Table 8-1). In addition, mine-life per ton costs of chromium from these placers appear to be comparable to the podiform deposits and 50% higher than Stillwater (table 8-6). The Maryland placer deposits are estimated to be considerably smaller than those of the Oregon beach sands, probably more costly to exploit and have a shorter mine life; therefore, they are more than likely not amenable to economic investment.

In spite of the apparent near-term noneconomic structure of U.S. chromite resources the U.S. Bureau of Mines has introduced a chromite beneficiation program in the last few years that has provided encouraging results. For example, the Salt Lake City research center has been conducting pilot investigations on Stillwater Complex ore; the study provided a combined upgraded concentrate, that when ground and floated, achieved a 46% CR_2O_3 concentrate. This work can have an important impact on the beneficiation economics of the largest and most important chromite deposits in the United States. By itself, it cannot make the deposits competitive in today's market. In addition, the Albany Research Laboratory of U.S. Bureau of Mines has been exploring the recovery of chromite from the residue of laterites that are chemically processed to recover cobalt and nickel. These residues may contain from 0.5% to 2% elemental chromium; through a series of steps involving both physical and chemical treatments, a combined concentrate from the residue in these ores has a 2:1 chromium to iron ratio and contains 40% to 50% of the initially contained chromium. A third project, undertaken by the Spokane field office of U.S. Bureau of Mines, is evaluating the low-grade, disseminated and podiform deposits of California. These ores range from 3% to 10% CR_2O_3, and as suggested above, contain tonnages that can potentially be mined by open pit and underground methods. Indeed, on a small scale some of these ores were actually mined during World War II and had an average feed assay of 7% to 14% CR_2O_3. Preliminary results suggest that these podiform ores can be concentrated in the range of 37% to 45% CR_2O_3, and with improved gravity process techniques, a marketable concentrate might be produced if smelter facilities were located nearby and local steel markets were accessible to the product.

Nickel Analysis

The nickel ores that are potentially producable in the time-frame 1995 can be characterized as nickel-bearing laterites, massive and disseminated sulfide deposits, and lead sulfide ores in which nickel is present as a secondary metal or byproduct. The five following tables (Tables 8-7 to 8-11) provide most of the available nickel resource and cost information in the U.S. Bureau of Mines' Mineral Availability System file.

In the United States the identified nickel resources related to the nickel laterites are located almost exclusively in the northern California/southern Oregon laterite belt. The best known of these deposits is the Hanna Mine, Riddle, Oregon, nickeliferous laterite, which until April of 1982 was producing approximately 14,000 tons per year of nickel metal. The Riddle deposit has been in operation since 1974 and has an estimated resource base that may provide up to 20 more years of mine life. Nickel is the only

TABLE 8-7
NICKEL INFORMATION

State Deposit Type	Type[1] of Deposit	Status[2]	Owner/Operator	Past Nickel Production	Proposed Mining Method	Proposed Processing Method	No. of Pre-Prod Years	Proposed Mine Life Years	Commodities
Alaska									
Brady Glacier	Sulf/Diss	Exp	Inspiration Devel. Co.	None	Open Stope	Flotation	9	11	Cu-Ni
Yakobi Island	Sulf/Diss	Exp	Inspiration Devel. Co.	None	Bench (BERM)	Flotation	5	17	Cu-Ni
California									
Lower Elk Camp Area[3]	Lat/Repl	Exp	California Nickel Corp.	None	Open Pit	Roast-Reduction Leach			Ni-Co
Gasquet	Lat/Repl	Exp	California Nickel Corp.	None	Open Pit	Roast-Reduction Leach	7	21	Ni-Co
Pine Flat Mtn.	Lat/Repl	Exp	Hanna Mining Co.	None	Strip Hillside	Roast-Reduction Leach	2	9	Ni-Co
Rattlesnake Mtn.[3]	Lat/Repl	Exp	Del Norte Mining Co.	None	Open Pit	Roast-Reduction Leach	3	14	Ni-Co
Red Mtn. Area[3]	Lat/Repl	Exp	Coastal Mining Co.	None	Open Pit	Roast-Reduction Leach	2	14	Ni-Co
Maine									
Crawford Pond	Sulf/Diss	Exp	Know Mining Corp.	None	Open Pit	Flotation	2	33	Cu-Ni
Minnesota									
Birch Lake	Sulf/Diss	Exp	INCO U.S. Inc/ Hanna Mining/ Duval Corp.	None	Open Pit	Flotation	5	25	Cu-Ni
Dunka River	Sulf/Diss	Exp	AMAX Expl-Inc.	None	Open Pit	Flotation	5	26	Cu-Ni
Ely Spruce	Sulf/Diss	Exp	INCo	None	Open Pit	Flotation	3	22	Cu-Ni
Minnamax	Sulf/Diss	Exp	AMAX	None	Sublevel	Flotation	3	41	Cu-Ni
Partridge R.	Sulf/Diss	Exp	U.S. Steel	None	Open Pit	Flotation	5	20	Cu-Ni

TABLE 8-7
NICKEL INFORMATION
(continued)

State Deposit Type	Type[1] of Deposit	Status[2]	Owner/Operator	Past Nickel Production	Proposed Mining Method	Proposed Processing Method	No. of Pre-Prod Years	Proposed Mine Life Years	Commodities
Missouri									
Buick Mine	Sulf/Rep	Prd	AMAX Lead Co./ Homestake/AMAX	None	Overhand Square Set	Flotation	1	27	Pb-Zn
Fletcher	Sulf/Rep	Prd	St. Joe Minerals	None	Overhand Square Set	Flotation	1	39	Pb-Zn
Madison	Sulf/Rep	Ppd	Anschutz	1944-1959	Room and Pillar	Flotation	3	10	Co-Ni
Magmont	Sulf/Rep	Prd	Cominco Am. Inc. /Dresser Minerals	None	Overhand Square Set	Flotation	1	39	Pb-Zn
Oregon									
Nickel Mt. Mine	Lat/Repl	Prd	Hanna Mining Corp.	1974-1982	Strip Hillside	Roast-Reduction Leach	0	20	Ni
Red Flats	Lat/Repl	Exp	Hanna Mining Corp./Red Flats Ni/Big Basin Nickel	None	Strip Hillside	Roast-Reduction Leach	4	7	Ni-Co
Eight Dollar Area:[3]									
Eight $ Mtn.	Lat/Repl	Exp	California Nickel Corp.	None	Open Pit	Roast-Reduction Leach	2	8	Ni-Co
Rough & Ready	Lat/Repl	Exp	Inspiration Devel. Co.	None	Open Pit	Roast-Reduction Leach	2	4	Ni-Co
Woodcock Mtn.	Lat/Repl	Exp	Coastal Mining Co.	None	Open Pit	Roast-Reduction Leach	2	5	Ni-Co
Washington									
Cle Elum[3]	Lat/Repl	Exp	Burlington Northern, Inc.	None	Open Pit	Roast-Reduction Leach	NA	NA	Ni-Co

TABLE 8-7
NICKEL INFORMATION
(continued)

[1] Type of Deposit:
  Sulf -- sulfide
  Diss -- disseminated
  Lat -- laterite
  Repl -- replacement
  NA -- not available

[2] Status:
  Dev -- developing mine
  Exp -- explored deposit
  Ppd -- past producer
  Prd -- producing mine

[3] Not costed at the 75% probability level

TABLE 8-8
NICKEL RESOURCES
(update)

Deposit Type Range TPY Ore	No. of Deposits	DEMONSTRATED RESOURCES					INFERRED RESOURCES					
		WT-PCT		MT			WT-PCT			MT		
		In Situ Ni Grade WT. AVG(%)	Range	In Situ Tonnage	Contained Ni Metal Content	Recoverable Ni Metal Content	No. of Deposits	In Situ Ni Grade Wt. AVG(%)	Range	In Situ Tonnage Content	Contained Ni Metal Content	Recoverable Ni Metal
Small 0-1,000,000	4	0.571	.256-.91	50,350,000	287,400	132,700	3	.572	.390-.840	27,360,000	156,500	126,200
Medium 1,000,001-10,000,000	8	0.439	.035-.90	686,614,000	3,013,600	2,462,300	6	.687	.021-.980	85,570,000	588,000	528,200
Large 10,000,001-20,000,000	4	.188	.16-.20	3,360,100,000	6,312,200	4,284,600	-	-	-	-	-	-

W -- Withheld due to confidentiality

TABLE 8-9
NICKEL RESOURCES
(update)

		DEMONSTRATED RESOURCES					INFERRED RESOURCES					
		WT-PCT		MT			WT-PCT			MT		
Deposit Type	No. of Deposits	In Situ Ni Grade WT. AVG(%)	Range	In Situ Tonnage	Contained Ni Metal Content	Recoverable Ni Metal Content	No. of Deposits	In Situ Ni Grade WT. AVG(%)	Range	In Situ Tonnage	Contained Ni Metal Content	Recoverable Ni Metal Content
Lead Sulfides	3	0.131	.021-.39	134,070,000	175,100	30,800	1	.191	.021-.39	14,110,000	26,900	4,700
Nickel Sulfides	9	0.217	.16-.51	3,835,000,000	8,308,900	5,773,900	-	-	-	-	-	-
Laterites	4	.867	.8-.9	134,300,000	1,165,000	1,075,000	7	0.726	.55-.98	98,821,000	717,600	649,700

W -- Withheld due to confidentiality

207F

TABLE 8-10
NICKEL COSTS

Type & Size Range TPY Ore	No of Deposits	OPERATING COST $/MT ORE Average Mine	Range	Average Mill	Range	Average Total	Range	CAPITAL COSTS MILLIONS Average Total	Range	AVERAGE ANNUAL PRODUCTION MT. Ore	MT. Recoverable Nickel
Small 0-1,000,000	4	10.40	4.60-16.80	18.20	4.70-50.90	28.50	9.30-59.50	68	9-169	590,000	1,905
Medium 1,000,001-10,000,000	8	8.50	2.70-13.50	13.40	4.20-36.60	22.10	7.20-47.50	231	16-377	2,300,000	7,214
Large 10,000,001-20,000,000 sulfides	4	W	W	4.15	4.00-4.60	W	W	437	367-470	13,930,000	17,273

#0030A

TABLE 8-11
NICKEL COSTS

Type of Deposit	No of Properties	OPERATING COST $/MT ORE						CAPITAL COSTS MILLIONS $		AVERAGE ANNUAL PRODUCTION	
		Average Mine	Range	Average Mill	Range	Average Total	Range	Average Total	Range	MT. Ore	MT. Recoverable Nickel
Lead Sulfides	3	7.20	4.60-8.60	4.70	4.50-4.90	11.90	9.30-13.20	17	9-22	1,340,000	700
Nickel Sulfides	9	8.30	2.90-16.80	5.70	4.00-7.50	14.00	6.90-23.10	294	17-470	7,344,000	11,723
Laterites	4	6.20	2.70-8.60	32.70	4.50-50.90	39.40	7.20-59.50	294	16-691	1,349,000	10,645

economic recoverable commodity at the mine. Other Oregon depostis of the laterite type are considerably small and lower in grade than Riddle ores, and usually have a very smaller amount associated cobalt.

The California laterites are also nickel-bearing with subsidiary cobalt, and contain resource tonnages at some sites that could provide a 10 to 20 year mine life. All of these deposits would be open pit or hillside strip mines, and on a combined basis, the demonstrated resources of four of the deposits evaluated by the U.S. Bureau of Mines have an in-situ average nickel grade of 0.87% and a combined tonnage range of 125 to 150 million tons containing over a million metric tons of nickel metal (see Table 8-11). The inferred resources of seven more deposits contain an in-situ average nickel grade of 0.73%, ranging from 0.55% to 0.98%. The estimated inferred resources are close to 100 million tons of ore containing 600,000 to 700,000 metric tons recoverable nickel. These resources, though not as large as those contained in the massive nickel sulfide deposits discussed below, still provide about 20% of the contained nickel metal in identified U.S. resources. They are, therefore, important in the assessment of long-range nickel mining in the United States.

These nickel laterites are amenable to mining in medium size operations ranging from 1 to 10 million tons per year ore. Some deposits, of course, might be mined in a much smaller physical plant, however, in terms of the demonstrated resource base nearly one-half the recoverable nickel can be produced in mid-sized mines of the 1 to 10 million tons per year category. Of the six deposits, that size category representing inferred resources, average ore grades were considerably lower, the in-situ tonnage an order of magnitude less, and only a relatively small amount of contained nickel could possibly be produced (see Table 8-8).

Most of the laterite resource base in the U.S. has been identified and awaits the economic and technological conditions that may make it amenable to mining. Average mine operating costs for the laterites are about $6.00 per metric ton, ranging from $3.00 to $8.50 depending upon the size of the deposit and the ability to take advantage of economies of scale (see Table 8-11). Average mill operating costs for the laterites are the highest for all nickel ores and amount to $33.00 per metric ton, ranging from $5.00 to $50.00 per ton. Combined operating costs of mine and mill for the nickel laterites average about $40.00 per ton, or about $2.25 per pound of contained nickel, and range from $7.00 to $60.00. Capital costs of these operations average $300 million and can be an order of magnitude less to as much as two times greater depending upon size, location, and applicable technologies. Average mine and mill operating costs alone are about equal to half the current value of nickel in a ton of laterite ore.

More specifically, USBM data in Tables 8-7 to 8-13 show that the mine-life operating and capital costs per pound of nickel in laterite deposits are about $2.50 per pound, compared to $4.00 per pound for nickel sulfide ores and over $10.00 per pound for lead sulfides (assuming all operating and capital costs are borne by nickel). The mining of laterites in small- to medium-size deposits also confirms that mine-life and operating and capital costs are close to the current selling price per pound of nickel (table 8-12). For the mine operator, additional operating and capital costs are search costs, administration, transportation, and other costs associated with a mining operation that are not included in this analysis. The selling price also includes profit; we assume, therefore, that an addition of 35% to the per

TABLE 8-12
NICKEL COST ANALYSIS
BY TYPE GEOLOGIC DEPOSIT*

	OPERATING COST				CAPITAL COST		
deposit type	MT[1] ore	grade[2]	$/MT Ni	$/annual Ni prod. MT	Incl.[3] $ cost & payback	assume[4] mine life	$/mine life
lead sulfides[5]	11.90	.000522	$22,780	$24,286	$4,857	29 yrs.	$167
Ni sulfides	14.00	.00160	$8,770	$25,079	$5,016	24 yrs.	$209
laterites	39.40	.00789	$4,993	$27,619	$5,524	10 yrs.	$552

TOTAL COSTS

	operating cost	capital cost	Total/MT	Total/LB	ratio
Pb Sulfides[5]	$22,780	167	$22,947	$10.43	4.1
Ni Sulfides	$8,770	209	$8,879	$4.08	1.6
Laterites	$4,993	552	$5,545	$2.52	1

---

*Based upon USBM data, 1982.

[1] From USBM, 1982 data.

[2] $\frac{\text{recoverable metal (annual prod.)}}{\text{ore (annual prod.)}}$

[3] assumed cost money and payback time 0.2.

[4] average projected mine life for all operations of particular type as listed by USBM.

[5] In reality, these costs would be shared by byproduct Co. production.

TABLE 8-13
NICKEL COST ANALYSIS
BY SIZE DEPOSIT*

	OPERATING COST			CAPITAL COST			
size deposit	$ MT ore	grade	$/MT Ni	$/annual Ni prod. MT	Incl. $ cost & payback	mine life	$/mine life
small[1]	28.50	.00323	$8,824	$35,696	$7,139	6 yrs.	$1190
medium[2]	22.10	.00314	$7,038	$32,021	$6,404	22 yrs.	$291
large[3]	W	.00124	--	$25,300	$5,060	24 yrs.	$211

TOTAL COSTS

	operating cost	capital cost	Total/MT	Total/LB
Small	$8,824	$1,190	$10,014	$4.55
Medium	$7,038	$291	$7,329	$3.33
Large	--	$211	---	--

---

*Based upon USBM 1982 data.

[1] all laterite operations.

[2] laterite deposits and all non-Minnesota sulfide deposits.

[3] all Minnesota Cu/Ni sulfide deposits.

pound cost of nickel in Tables 8-12 and 8-13 should approximate a selling price for the laterites in the U.S. This price should be close to competitive with the world price; however, the costs of mining smaller deposits, transportation, and other costs and reasonable profit structure are not yet part of our analyses because mining is not in force today. Although the analysis can be considered soft, it at least gives the most reasonable set of conclusions that can be drawn without specific mine data. Also, our analysis suggests the desirable sequence of nickel mining addition on a comparative cost basis in the U.S. (i.e., laterites are more economic than nickel sulfides and medium-size deposits (and mines) are preferred over small deposits).

The massive nickel sulfide deposits of the United States that have been evaluated by the U.S. Bureau of Mines are characterized by the northern Minnesota massive and disseminated sulfides at Minnamax, Ely Spruce, Dunka River, Birch Lake and Patridge River, and have an in-situ ore grade averaging about 0.12% nickel. In addition, the combined demonstrated ore has been estimated to be well over 4 billion tons, containing at least 6 million metric tons of recoverable nickel metal (see Table 8-9). The inferred resource base for this class of nickel deposits has not been estimated by the U.S. Bureau of Mines. By extrapolation, the characterization of the in-situ resources associated with the Minnesota disseminated massive sulfide deposits can also be applied to the Alaskan Brady Glacier and Yakobi Island ores. They are smaller in size and contained ore tonnage, of low nickel grade, and also have associated copper as the primary metal for extraction. Reliable resource evaluations for the Alaskan deposits have not yet been made.

The ores of the nickel sulfide type will probably only be extracted from very large bulk mining operations, taking advantage of the economies of scale necessary in such low-grade environments. The amount of possible recoverable nickel in the four largest of these potential mines amounts to 80% of the total recoverable nickel of all the nickel sulfide deposits known in the U.S., and would exceed 4 million metric tons of metal (Table 8-10).

The cost of mining nickel sulfides (Table 8-11) averages $8.00 per ton for mine operating costs and ranges from $3.00 to $17.00 per metric ton. Mill costs average $6.00 per ton with a range of $4.00 to $7.50 per metric ton. Total average operating costs for the nickel sulfide deposits will be about $14.00 per ton, or about $3.90 per pound of contained nickel, with a range of $7.00 to $23.00 per metric ton, considerably less than the laterites. At the low cost operating range, nickel laterites and massive nickel sulfide deposits are perhaps comparable on a per ton basis; capital costs are also at the same order of magnitude averaging about $300 million and ranging from an order of magnitude less, to up to twice that amount for the installation of a mine and beneficiation system in a large operation. The average annual production of ores from the nickel sulfide properties in the Duluth Complex could provide anywhere from 10% to 50% of the annual U.S. consumption of nickel in 1995 if economic incentives were available. At current price, however, the nickel value in a ton of Minnesota sulfide ore is only about 60% of the price needed to encourage mining if all costs and values are attributed to the nickel in the ore.

The last class of potential nickel ores that might be exploited in the United States, within the time frame of interest, are the lead ores of the East Missouri district, which contain accessory amounts of nickel. Only with cobalt, however, might the nickel be economically and readily recovered. In this case nickel would not be the primary ore, and the total demonstrated resource base contains only a very small amount of nickel metal, estimated by the U.S. Bureau of Mines to be 175,000 metric tons for the three deposits studied (Table 8-9); these deposits have an average nickel grade of 0.13%; however, only about one-fifth of the contained nickel of these ores is thought to be recoverable. Within the inferred category only one deposit was evaluated by the U.S. Bureau of Mines and it contains a small amount of recoverable metal, estimated to be less than 5,000 metric tons (Table 8-9). As a byproduct of lead mining and cobalt recovery, nickel recovery depends upon nickel and cobalt price and the amount of lead mining. Zinc and copper are also important recoverable metals in these ores but in small quantities.

When all costs are attributed to nickel in the mining of lead sulfide, mine operating costs for nickel recovery are estimated to average $7.00 per metric ton of ore, ranging from $4.50 to $8.50 per ton; and mill costs are estimated to average $4.70 a ton, ranging from $4.50 to $5.00 a ton. Total average operating costs would be approximately $12.00 a ton and along with an average capital investment of $15 to $20 million (Table 8-11) will not produce an ore that can be mined primarily for its nickel value. Table 8-12 provides a min-life cost comparison of the lead sulfide, nickel sulfide, and laterite ores. Clearly, with all mining and capital costs charged against nickel in the lead sulfides, this ore is 2.5 to 4.0 times more expensive than the alternatives. The decision to recover these nickel values, however, will probably hinge on the desire to recover the associated cobalt. Within the 1995 time-frame, the U.S. Bureau of Mines estimates that 1.5 million pounds per year of nickel production is possible technically (Table 8-11).

The current price of nickel in the world market is approximately $3.50 a pound and does not provide an adequate incentive to mine any of the above discussed nickel deposits. According to our analysis, the price of nickel (in 1982 dollars) is just about what is needed to mine the nickel laterites of Oregon and California; the price will have to increase by at least 49% to bring the nickel sulfide ores on line for production, and increase by threefold if the cost of mining the East Missouri lead ores are to be paid by nickel values alone. Low or falling ore grades, declining or small reserves, the cost of energy, and poor market outlook, at least for the short term, have inhibited the mining of Riddle, Oregon, and other like laterites in Oregon and California. It is very unlikely that the copper-nickel sulfide ores of Minnesota and Alaska could be mined for their metal content unless world nickel prices are almost 50% higher than the current level, or the price of copper is substantially higher, or both. Within the world nickel and copper market structure it is highly uncertain that real prices of this magnitude will be reached by the year 1995 and remain there, given the current world excess capacity of nickel and copper production. Therefore, within the next 12 years, the only primary nickel ores that might be exploited are the nickel laterites and then perhaps only at the Riddle deposit, richest of the known deposits and the byproduct recovery from lead sulfide ores of east Missouri, provided cobalt is recovered from the ore. In this case, nickel is probably localized in the iron sulfides along with cobalt, although it is not clear at this time that all of the nickel resides within the iron sulfide fraction. Further study must be done in order to evaluate the full potential of these ores, even though they may provide less than 5% of the annual nickel requirements of the United States.

COBALT ANALYSIS

Economic cobalt production in the United States will depend primarily on

(1) cobalt recovery as a byproduct from ores that are primarily mined for copper, nickel, or lead, and (2) as primary cobalt ores of the Blackbird Mine in central Idaho. The copper/cobalt ores are characterized by occurrence in ultramafic layered intrusives like the Ely Spruce and Minnamax deposits of the Duluth Complex; the nickel/cobalt ores are the laterites of California and Oregon; and the cobalt-rich sulfide ores of Lead Belt mineralization in eastern Missouri. With the exception of the nickel/cobalt laterites of northern California and southern Oregon, mining from these resources would be mainly by underground systems.

The U.S. Bureau of Mines assessment of the demonstrated and inferred cobalt resources of these four types of geologic occurrences provide the following information (Tables 8-14 to 8-19). The Blackbird ores are primarily cobalt arsenides and average about 0.75% cobalt in the ore. Other metals, especially copper, are also present, but cobalt would be the primary metal recovered from any mining operation. Dames and Moore (1980) suggest a demonstrated resource of 35,000 tons of cobalt metal and an inferred or possible resource of another 110,000 tons for Blackbird (see Chapter V). U.S. Bureau of Mines data on the Blackbird cobalt ore are not available to the public. Given the extent of the exploration in the Blackbird district, the structure of the ore zone (which may in fact be a stratabound metal sulfide occurrence rather than the vein type deposit), and the uniformity of grade distribution, the demonstrated and inferred resources as suggested above may actually be a minimum. The recovery rates of the Blackbird Mine could reach 3,500 to 4,000 tons of cobalt per year under maximum production (table 8-14). In addition it is important to note that only a small part of the Blackbird district and its extensions have been explored in any detail for cobalt mineralization. The district as a whole has a resource potential far

TABLE 8-14A
POTENTIAL UNITED STATES COBALT PRODUCTION

Property	Owner	Cobalt Reserves (tons cobalt)		Estimated Maximum Recoverable Annual Production Cobalt (tons cobalt)	
Blackbird mine, Idaho	Noranda Mines Limited	Proven:	30,000	1,800	(initial)
		Possible:	110,000	3,830	(possible)[a]
Madison mine, Missouri	Anschutz Uranium Corporation	Possible:	22,000	9,000	
Gasquet Mountain mine, northern California	California Nickel Corporation	Probable & Possible	16,200	900	
Duluth Complex, Minnesota	AMAX Inc. Inco, Limited	Possible	85,000	600	

Source: Dames & Moore, 1980, p. 8.

[a] Dames & Moore, 1980, not explained

TABLE 8-14B
COBALT INFORMATION

State Deposit	Status[1]	Owner/Operator	Past Cobalt Production	Proposed Mining Method	Proposed Process	No. of Pre-Prod Years	Proposed Mine Life Years	Commodities
Alaska								
Yakobi Island	Exp	Inspiration Devel. Co.	None	Bench (BERM)	Flotation	5	17	Cu-Ni
California								
Gasquet	Exp	California Nickel Corp.	None	Open pit	Leach/PPT	7	21	Ni-Co
Grey Eagle	Ppd	Noranda	NA	Open pit	Flotation	2	23	Cu
Rattlesnake Mtn[2]	Exp	Del Norte Mining Co.	None	Open pit	Leach/PPT	3	14	Ni-Co
Lower Elk Camp[2]	Exp	California Nickel Corp.	None	Open pit	Leach/PPT			Ni-Co
Pine Flat Mtn	Exp	Hanna Mining Corp.	None	Strip Hillside	Hydrometallurgy	2	9	Ni-Co
Red Mountain[2]	Exp	Hanna Mining Corp.	None	Open pit	Leach/PPT	2	14	Ni-Co
Idaho								
Blackbird Mine	Dev	Noranda	None	Horizontal Cut & Fill	Hydrometallurgy	2	12	Co-Cu
Minnesota								
Ely Spruce	Exp	International Nickel Co.	None	Open pit	Flotation	3	22	Cu-Ni
Minnamax	Exp	AMAX, Inc.	None	Sublevel	Flotation	3	41	Cu-Ni
Oregon								
Red Flat	Exp	Hanna Mining/Red Flat Ni/ Big Basin Ni	None	Strip Hillside	Leach/PPT	4	7	Ni-Co
Eight Dollar Area:								
Eight $ Mtn	Exp	California Nickel Corp.	None	Open pit	Leach/PPT	2	8	Ni-Co
Rough & Ready	Exp	Inspiration Devel. Corp.	None	Open pit	Leach/PPT	2	4	Ni-Co
Woodcock	Exp	Coastal Mining Corp.	None	Open pit	Leach/PPT	2	5	Ni-Co

Source: USBM, MAS File, 1982.

TABLE 8-14B
COBALT INFORMATION
(continued)

State Deposit	Status[1]	Owner/Operator	Past Cobalt Production	Proposed Mining Method	Proposed Process	No. of Pre-Prod Years	Proposed Mine Life Years	Commodities
Missouri								
Boss-Bixby	Exp	Getty Oil Co/Azcon Corp/ Hanna Mn	None	Sublevel	Flotation	18	28	Cu-Co-Fe
Brushy Creek	Prd	St. Joe Minerals Corp.	None	Overhand Square Set	Flotation	1	45	Pb
Buick	Prd	AMAX and Homestake	None	Overhand Square Set	Flotation	1	27	Pb-Zn
Viburnum #28 and #29	Prd	St. Joe Minerals Corp.	None	Overhand Square Set	Flotation	1	36	Pb-Zn
Fletcher Div	Prd	St. Joe Minerals Corp.	None	Overhand Square Set	Flotation	1	39	Pb-Zn
Frank Milliken	Prd	Ozark Lead (sub of Kennecott)	None	Overhand Square Set	Flotation	1	22	Pb-Zn
Higdon	Dev	Bunker Hill Mining Co.	None	Overhand Square Set	Flotation	1	14	Pb-Zn
Madison Mine	Ppd	Anschutz Uranium Corp.	1944-1959	Room & Pillar	Flotation	3	10	Co-Ni
Magmont	Prd	Cominco Am and Dresser	None	Overhand Square Set	Flotation	1	39	Pb-Zn
West Fork	Exp	Asarco	None	Horizontal Cut & Fill	Hydrometallurgy	2	31	Pb

---

[1] Status: Dev - developing mines, Exp - explored deposit, ppd - past producer, prd - producing mine.

[2] Not costed at the 75% probability level.

TABLE 8-15
COBALT RESOURCES

Deposit Type	No. of Deposits	DEMONSTRATED RESOURCES					INFERRED RESOURCES					
		WT-PCT		MT				WT-PCT		MT		
		In Situ Co Grade WT. AVG(%)	Range	In Situ Tonnage	Contained Co Metal Content	Recoverable Co Metal Content	No. of Deposits	In Situ Co Grade WT. AVG(%)	Range	In Situ Tonnage	Contained Co Metal Content	Recoverable Co Metal Content
Lead-Zinc	8	0.015	.005-.180	310,390,000	48,600	7,300	6	0.012	.005-.014	44,020,000	5,400	700
Nickel-Cobalt Laterites	3	0.104	.032-.150	103,990,000	53,800	44,000	6	0.068	.032-.090	93,150,000	60,700	50,700
Primary Cobalt	2	W	W	W	W	W	0	-	-	-	-	-
Copper-Cobalt	5	0.023	.017-.080	704,100,000	156,600	78,000	1	.08	.08-.08	5,000,000	4,000	1,800

W -- Withheld due to confidentiality.

Source: USBM, MAS File, 1982.

344E

TABLE 8-16
COBALT RESOURCES

Deposit Type (Tpy. Ore Prod.)	DEMONSTRATED RESOURCES						INFERRED RESOURCES					
		WT-PCT			MT			WT-PT			MT	
	No. of Deposits	In Situ Co Grade WT. AVG (%)	Range	In Situ Tonnage	Contained Co Metal Content	Recoverable Co Metal Content	No. of Deposits	In Situ Co Grade WT. AVG (%)	Range	In Situ Tonnage	Contained Co Metal Content	Recoverable Co Metal Content
Missouri Lead Deposits												
Small 0-1,000,000	3	0.03	.01-.18	53,070,000	16,800	1,900	2	.03	.01-.03	8,590,000	2,200	200
Large 1,-15,000,000	5	.01	.005-.025	257,320,000	31,800	5,400	4	.01	.005-.014	35,430,000	3,200	700
Remaining Deposits												
Small 0-1,000,000	4	.02	.080-.740	25,920,000	59,100	44,200	2	.08	.075-.080	20,202,000	15,400	13,100
Large 1,-15,000,000	6	.02	.017-.15	792,310,000	197,600	114,100	4	.06	.032-.081	77,950,000	49,300	39,400

W--Withheld due to confidentiality

Source: USBM MAS File, 1982.

#0030A

TABLE 8-17
COBALT COSTS

Deposit Type	No of Deposits	OPERATING COSTS				$/MT ORE		CAPITAL COSTS MILLIONS $		AVERAGE ANNUAL PRODUCTION	
		Average Mine	Range	Average Mill	Range	Total	Range	Average	Range	Ore MT	Recoverable Cobalt MT
Lead-Zinc[1]	8			.30	.10-.70	.30	.10-.70	1,180	430-4,400	1,200,000	1,000
Nickel-Cobalt Laterites[2]	4	6.00	4.90-8.70	35.40	28.20-46.60	41.40	33.10-55.30	283,000	170,000-346,000	1,349,000	927
Primary Cobalt	2	W	W	W	W	W	W	W	W	W	W
Copper-Cobalt	5	10.50	2.90-15.60	5.20	4.20-5.90	15.70	7.50-20.90	150,000	15,000-377,000	4,050,000	400

W - Withheld due to confidentiality

[1] These figures reflect incremental costs to produce cobalt as a byproduct of lead-zinc.

[2] Eight $ Mtn, Rough and Ready, and Woodcock evaluated as one property in cost information.

Source: USBM MAS File, 1982.

Table 8-18
Cobalt Cost Analysis

Operating Cost

Deposit Type	$MT ore	grade	$MT Co	$LB Co
Pb/Zn	$.30	.00083	$361	$.16
Laterites	$41.40	.00069	$60,000	$27
Primary Co	W	W	--	
Cu/Co	$15.70	.0001	$157,000	$71

Capital Cost

$/Annual Co Prod MT	Inc. Cost & Payback	Assumed Mine Life	$/MT Total Prod	$/LB Total Prod
$1180	$236	32 yrs	$7.40	$0
$305,286	$61,057	12 yrs	$5088	$2.31
W	--	11 yrs	--	
$375,000	$75,000	27 yrs	$2778	$1.26

TOTAL COSTS

	Operating Cost	Capital Cost	Total
Pb/Zn	.16	0	.16
Laterites	$27	$2	$29
Primary Co	--		
Cu/Co	$71	$1	$72

Note: Actual Co. Selling price

1977	5.60/lb
1978	11.50/lb
1979-80	25.00/lb

Source: USBM, 1982, MAS Files, Tables 8-14 to 8-17.

TABLE 8-19
MANGANESE INFORMATION

State Deposit	Status^2	Owner/Operator	Past Production	Proposed Mining Method	Proposed Milling Method	No. of Pre-Prod Years	Proposed Mine Life Years
ARIZONA							
Hardshell	Explored	N/A	None	Room and Pillar	Leach/PPT	4	10
Maggie Mine	Part Dev.	Ariz. Manganese Corp.	None	Room and Pillar	Leach/PPT	6	26
COLORADO							
Sunnyside Mine	Developed	Standard Metals Corp.	None	Overhand Shrinkage	Leach/PPT	4	20
MAINE							
Maple/Hovey	Exp. Pros.	Consortium	None	Strip Level	Leach/PPT	6	61
North Aroostook Dist.	Exp. Pros	Corsortium	None	Open Pit	Leach/PPT	6	25
MINNESOTA							
Cuyuna North Range	Past Prod.	Private Land	1912-N/A	Open Pit	Leach/PPT	4	14
MONTANA							
Butte District	Past Prod.	Anaconda	1917-1959	Horizontal Cut and Fill	Flotation	3	3
NEVADA							
Three Kids District	Past Prod.	J.C. and Fern Lancaster	1917-1961	Open Pit	Flotation	3	7

Source: USBM, MAS File, 1982.

larger than the figures for the Blackbird mine itself.

Of the five copper/cobalt demonstrated deposits evaluated by U.S. Bureau of Mines (Table 8-15) the average cobalt grade is 0.023% ranging from 0.017% to 0.08%. The demonstrated in-situ resources in the five deposits is approximately 700 million tons, containing about 150,000 tons of cobalt and 75,000 tons of recoverable cobalt metal. The discrepancy between total metal content of the ore and recoverable metal relates primarily to the present efficiency and cost of cobalt recovery from copper sulfide ores. Future advances in this area are indeed probable, but the 1995 target date assumes current capability.

The nickel/cobalt laterites of California and Oregon, on the basis of an evaluation of three demonstrated deposits by the U.S. Bureau of Mines (Table 8-16) contain an average cobalt grade of 0.10% with a range of 0.03% to 0.15%. The in-situ demonstrated resource is estimated to be 100 million tons of ore containing about 50 to 55 thousand tons of metal, of which 45,000 tons are recoverable. The inferred resources of six laterite deposits contain an average grade considerably lower than the demonstrated resources, estimated to average 0.07% cobalt with a range of 0.03% to 0.09%. The estimated in-situ tonnage is comparable to the demonstrated resources and is between 90 to 100 million tons of ore containing about 50,000 tons of recoverable nickel metal. Cobalt/nickel laterites, therefore, are comparable in potential metal supply with Blackbird, Idaho, and the Duluth Complex of Minnesota.

Cobalt recovery from the east Missouri Lead Belt, on the basis of the evaluation of eight deposits by the U.S. Bureau of Mines (Table 8-16), will provide an order of magnitude less cobalt than the other three ore types. The average cobalt grade of demonstrated resources is about 0.015%, ranging from

0.005% to 0.18% cobalt; an in-situ demonstrated resource estimate of 300 million tons of ore is available in the east Missouri Belt; however, the contained cobalt metal content of 50,000 tons is only recoverable at 15% of the amount present, using present technology (Table 8-16). The inferred resource category for these cobalt-bearing lead ores contains six deposits having a total estimated tonnage of 40 to 45 million tons, and at approximately the same ore grade and would only yield only about 700 tons of recoverable cobalt metal.

On the basis of mine size alone, excluding the east Missouri Lead Belt deposits, the laterite and cobalt sulfide ores would probably be most economically recovered in operations producing 1 to 15 million tons per year of ore. The U.S. Bureau of Mines estimates that about 35% to 40% of the total recoverable nickel in these ores would come from small deposits (up to 1 million tpy), while 60% to 65% of the nickel would come from mining operations in the 1 to 15 million ton a year category (Table 8-17). This distribution relates primarily to the economics of mining in the Duluth Complex and perhaps some of the larger nickel/cobalt laterites of California and Oregon. Small tonnage development would take place in the Blackbird mining district and most of the nickel/cobalt laterites of California and Oregon.

With the exception of the Blackbird deposit, cobalt mining costs, have also been presented by the U.S. Bureau of Mines (Table 8-17). For the Missouri Lead Belt, mining costs have been and will be expended as a function of lead recovery. Additional mill operating costs would average about $.30 per ton, with a range of between $.10 to $.70 per ton in 1982 dollars. The average capital costs for recovering these ores would average about $1 to $2 million and be in the range of $.50 to $5.0 million. Average annual production from this source could reach a million tons of ore per year, producing two million pounds (1000 tons) of cobalt.

The cobalt/nickel laterites, on the basis of an evaluation of four deposits, have mine operating costs averaging $6.00 a ton, and ranging from $5.00 to $8.70 in 1982 dollars (Table 8-18). Average mill costs are approximately $35.00 a ton, ranging from $28.00 to $46.00 per ton. Total operating mine and mill costs, therefore, average $41.00 per ton and range from $33.00 to $55.00. Capital investment costs for these large cobalt/nickel laterites would approach $300 million and range from $150 to $350 million. Average mine size could produce 1.3 million tons of ore per year containing about 2 million pounds of cobalt and providing a production capability approximately equal to the Missouri Lead Belt.

The Duluth Complex copper/cobalt ores, based on the evaluation of five deposits by the U.S. Bureau of Mines (Table 8-18), would have average mine operating costs of $10.50 a ton, with a range of $3.00 to $15.00 a ton in 1982 dollars. Average mill costs of $5.00 a ton, ranging from between $4.00 to $6.00 ton, have also been projected. In total the copper/cobalt Duluth Complex ore has an estimated average operating cost of $16.00 a ton, with a range from between $7.00 to $20.00 a ton. Average capital costs for these deposits would be approximately $150 million, ranging from a low of $15 million to a high of $400 million. Because these ores are so low grade, however, an annual production of 4 million tons of ore per year would only produce 500 to 800 tons of cobalt metal.

Data on costs for the primary cobalt deposits, specifically mining, milling, and metallurgical charges, are not available for the Blackbird Mine. We do know, however, that Noranda Mining Company has proposed a base price subsidy of $20.00 a pound that would cover mining and processing costs and provide a return on the investment adequate for the company. With ore grades averaging about 0.75% cobalt and 1.5% copper at Blackbird, and demonstrated resources of 35,000 tons of cobalt metal, tonnages large enough to warrant a significant investment in mine refurbishment and expansion, mill renovation and new metallurgical facilities, it is clear that the $20.00 a pound 1982 base is the minimum price at which U.S. cobalt resources can be available in the United States.

Mine-life costs of cobalt, one a per pound basis, for the four types of U.S. cobalt deposits, are given in table 8-19. The costs for east Missouri Pb Belt ores are clearly incremental costs associated with cobalt recovery only after lead and other metals have paid the bulk of mine capital and operating expenses. It would appear that cobalt recovery as an add-on to the present operation is indeed presently economic, provided that smelter facilities for the ore ore is available, or can be constructed at costs that maintain the economic viability of the operation.

Recovery of cobalt from the laterite ores appear to be more economic by a factor of at least two to 2 1/2 over cobalt from massive Cn/Ni/Co ores. In addition, for cobalt recovery from laterite to approach economic targets of recovery, the cobalt price in 1982 dollars must rise to slightly above its recent all-time high, relative to the current spot market for cobalt, this means an increase of about 5 times the current price. Should cobalt approach $30.00 per pound in the next 10 years, a price which appears likely only with cut-off supply from the worlds supply centers could the nickel laterites appear attractive for cobalt recovery.

Even further removed, with the need for the primary copper and nickel, would be the cobalt contained in the massive sulfide ores. These deposits have an extremely low likelihood of production by the 1995 target date. The primary cobalt ores of Blackbird, have not been assessed in Table 8-18 due to lack of USBM public data. We do know, however, that they are the most economic of the four types presented, and would have the lowest mine-life costs considering cobalt to be the primary metal of recovery. Given the subsidy price requested by Noranda last year, the mine-life cost of cobalt from Blackbird might be close to $15.00 per pound.

The U.S Bureau of Mines, in a recent survey of domestic cobalt availability using the MAS analysis (RI 8848, 1980), suggests that at an average copper price of $1.00 a pound, a lead price of $.40 a pound, and a cobalt price of $25.00 a pound, the recoverable tonnage of cobalt in the United States approaches 87,000 metric tons. This amounts to a little less than a third of all the contained cobalt in the identified (demonstrated plus inferred) resource base of the United States (Table 8-16). At the prices indicated, 15 of the properties studied by U.S. Bureau of Mines could produce cobalt and expect a 15% rate of return. In addition, at an assumed cobalt price of $15.00 a pound, the recoverable cobalt tonnage declines to 45,000 metric tons.

This study, based on a static analysis with data compiled in 1979, does not give an up-to-date picture of the real capacity to produce cobalt in the U.S. Clearly, there is no cobalt that can be produced in the United States today at $15.00 a pound. Also, the amount of cobalt that could be produced at a price of $25.00 a pound is, in our judgment, more than likely to be that which can be mined from the Blackbird ore bodies and recovered from the east Missouri Lead Belt ores. Combining the cobalt producible in those two mining environments might give as much as 40,000 to 45,000 tons of cobalt at a minimum cut off of $25.00 a pound in 1982 dollars, rather than the 80,000 tons suggested by the U.S. Bureau of Mines.

The same study by the U.S. Bureau of Mines also suggested that at copper prices between $1.00 to $1.50 a pound cobalt recovery of 25,000 tons might ultimately be possible from the ores of the Duluth Complex. What is clear at this time is that the copper/cobalt, massive sulfide deposits of Minnesota and S.E. Alaska cannot be brought on line with copper prices of $1.00 to $1.25 a pound, and it is unlikely that#these ores can even be mined at copper prices much below the range of $1.50 to $2.00 a pound. Consequently, the anticipated price of cobalt is almost independent of the capability to produce cobalt metal from these Duluth ores, and the changing conditions of U.S. copper mining make it even less likely that the Duluth Complex or Alaskan ores will be exploited in the near future (i.e., 1995).

Based on 1979 data, the Bureau of Mines also suggests that the Missouri Lead Belt could produce 7,000 metric tons of cobalt at prices around $22.00 a pound. We know that the minimum cut off price, in 1982 dollars, for cobalt production in the Missouri Lead Belt is now $25.00 a pound; with a lead time of 2 to 3 years before productioncapability is established, it is more than likely that an actual price of $25.00 to $30.00 a pound will be required to bring cobalt from these mines on market by 1995.

MANGANESE ANALYSIS

Within the United States the largest demonstrated manganese resources of the sedimentary deposit type are the Maple/Hovey ores, north Aroostook County, Maine, and the Minnesota north Cuyuna district together containing over 300 million tons of ore, of which 17 million metric tons of manganese could be recovered (Tables 8-20, 8-21, and 8-22). The Maine deposits have an average grade of 9 weight percent Mn. The Minnesota Cuyuna deposits have an average ore grade of approximately 8% Mn, contain 50 million tons of ore, and 3 million metric tons of recoverable manganese. Considerably smaller amounts of manganese are also recoverable from the disseminated replacement deposits of Hardshell, Arizona, and the breccia filling and fissure veins of the Maggie Mine, also in Arizona. Fissure filling and shears at the Sunnyside deposit of Colorado also contain manganese ores. The Butte Mt. vein ores contain small amounts of manganese. Becoming more important on the periphery of the mineralized zone, these ores are no longer available since the shutdown of the Butte operations. The Three Kids manganese district of Nevada also contains sedimentary ores containing less than a million tons of recoverable manganese. In summary, then, inferred resources are dominated by the Cuyuna, Minnesota, deposit with an estimated in-situ occurrence of 100 million metric tons containing 6 million tons of recoverable metal.

All of these deposits, considering the grade of ores, are currently uneconomic. Production from these ores cannot take place without government subsidy or a major cost-reducing advancement in technology, or perhaps even both. The current world market price of manganese is about $1.70 per long ton unit; the U.S. ores that might be potentially recoverable in the year 1995 at prices between $5.00 and $10.00 per long ton unit, in 1981 dollars, is only about 100 to 150 thousand metric tons (figure 8-1). South African and other world manganese resources are so dominant with respect to economics and supply as to preclude the development of significant U.S. resources within the time frame of the 1995 scenario. The U.S. Bureau of Mines estimates, however, that at $10.00 per long ton unit, in 1981 dollars, as much as 400 to 500 thousand

251B

TABLE 8-20
MANGANESE RESOURCES

			DEMONSTRATED RESOURCES				INFERRED RESOURCES				ANNUAL PRODUCTION	
STATE	DEPOSIT NAME	DEPOSIT TYPE	In Situ Grade WT-PCT	In Situ Tonnage MT	Contained Metal Content MT	Recov. Metal Content MT	In Situ Grade WT-PCT	In Situ Tonnage MT	Contained Metal Content MT	Recov. Metal Content MT	Ore	Con-centrate
AZ	Hardshell	Disseminated Replacement	15.0	5,895,500	884,300	641,300					536,000	96,300
	Maggie Mine	Sedimentary Fissure Vein, Breccia Fill	8.75	8,441,000	738,600	500,000					328,000	29,500
	TOTAL		11.32	14,336,500	1,622,900	1,141,300						
CO	Sunnyside	Fissure Vein, Shear zone, Breccia Fill	10.0	24,909,000	2,490,900	1,245,500	10.0	15,477,000	1,547,700	773,850	635,000	47,500
ME	Maple/ Hovey	Sedimentary	8.87	260,000,000	23,062,000	13,837,200					4,263,000	339,300
	N. Aroostook District	Sedimentary	9.54	63,100,000	6,019,700	3,611,800					2,620,000	224,000
	TOTAL		9.00	323,100,000	29,081,700	17,499,000						
MN	Cuyuna No.	Sedimentary	7.84	48,960,000	3,838,500	3,070,800	8.3	92,772,000	7,700,100	6,160,100	3,570,000	327,500
MT	Butte District	Fissure Vein, Breccia Fill, Stockwork	18.0	1,232,000	221,800	199,900					400,000	221,500
NE	Three Kids District	Sedimentary	13.2	7,230,000	954,400	668,100	13.2	7,270,000	959,600	671,700	1,050,000	210,900
	GRAND TOTAL			419,767,500	38,210,200	23,824,600		115,519,000	10,207,400	7,605,600	13,402,000	1,496,500

Source: USBM, MAS File, 1982

TABLE 8-21
MANGANESE COSTS

	Range TPY Ore	OPERATING COSTS - $/MT Ore			CAPITAL COSTS - MILLIONS OF $		
		No. of Deposits Evaluated	WTD Avg.	Range	No. of Deposits	Avg	Range
Small	328,000-1,050,000	5/5	46.30	41.20-51.20	3/5	180	143-205
Large	2,620,000-4,263,000	3/3	31.70	28.70-34.60	3/3	738	686-824

Source: USBM, MAS File, 1982

Table 8-22
Manganese Costs

Operating Cost

Size Deposit	$MT ore	grade	$MT Mn
Small	$46.30	0.06	$772
Large	$31.70	0.07	$453

Capital Cost

	$/Annual Mn Prod MT	Inc. Cost & Payback	Assumed Mine Life	$/Total Mn Prod
Small	$6000	$1200	15 yrs	$80
Large	$5,271	$1,054	25 yrs	$42

TOTAL COSTS

	Operating	Capital	Total/MT	Total/LTU
Small	$772	$80	$852	$8.70
Large	$453	$42	$495	$5.00

Small 500,000 MTY/ore < 30,000 MTY/Mn

Large 2,000,000 MTY/ore < 140,000 MTY/Mn

1980 Price 1.50/LTU Mn

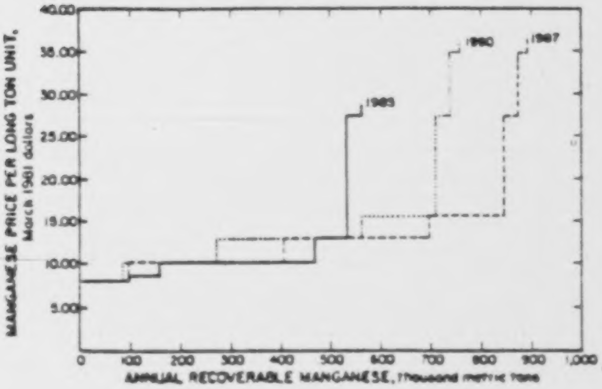


FIGURE 8-1 — ANNUAL PRODUCTION OF DOMESTIC MANGANESE FOR SELECTED YEARS AT VARIOUS PRICES.

Source: USBM, IC8889, 1982, p. 8.

25/D

metric tons.of recoverable manganese annually could be produced in the United States (Figure 8-1). For the amount of annual menganese production to rise significantly above that level, the price of manganese must climb to at least $35.00 a long ton unit, in 1981 dollars. There will be no U.S. terrestrial manganese production, therefore, in the 1995 time frame, and assuming manganese cost and price escalation at a rate of 2% to 3% a year, it is highly unlikely that any U.S. manganese would be mined without a substantial subsidy, at least to the year 2010.

The above manganese analysis is confirmed by mine-life costs as calculated from the USBM MAS file data (table 8-22) and presented in table 8-23. For the deposits evaluated by the USBM, it appears that large mine operations would have a $5.00 per L.T.U. mine-life cost, while small deposits mined by small miners, would show costs close to $9.00 per L.T.U. costs. Clearly, U.S. potential to produce mangganese at near the current price does not exist, and either a sharp run-up in price or a considerable subsidy, will be required to mine even 10% to 20% of our national demand by the year 1995.

PLATINUM ANALYSIS

The potentially or currently recoverable U.S. platinum resources are restricted to three known deposits, the Stillwater Complex of Montana, the Duluth Complex, Minnesota, and Goodnews Bay, Alaska. The U.S. Bureau of Mines evaluation of five unnamed U.S. platinum resources (Table 8-23) calculated a demonstrated resource base of 2.5 million troy ounces of platinum contained in 650 million tons of ore, with weighted average ore grade of 122 parts per billion. Inferred resources estimated in two unnamed U.S. deposits have a recoverable platinum content of 10 million troy ounces contained in 140 million tons of ore. The U.S. Geological Survey estimates total identified

TABLE 8-23
PLAINTUM RESOURCES[1]

	DEMONSTRATED RESOURCES					INFERRED RESOURCES					ANNUAL PRODUCTION	
	No. of Deposits	In Situ Grade WT-AVG (%)	In Situ Tonnage MT	Contained Platinum Content Tr. Oz	Recoverable Platinum Content Tr. Oz	No. of Deposits	In Situ Grade WT-AVG	In Situ Tonnage MT	Contained Platinum Content Tr. Oz	Recoverable Platinum Content Tr. Oz	Ore MT	Reco-verable Platinum Tr. Oz
Totals	5	.00002	660,170,000	3,309,000 (104 MT)	2,639,000 (83 MT)	2	.00028	140,712,000	12,4~~27~~72,000 (391 MT)	10,227,000 (321 MT)	19,967,000	110,800 (3.5 MT)

W - Withheld due to confidentiality

[1] All information concerning individual domestic platinum resources and costs is confidential.

Source: USBM, MAS File, 1982.

platinum resources in the U.S. at about 23 million troy ounces, of which 4.5 million troy ounces are in the Stillwater Complex and the rest in Alaska. Hypothetical and speculative platinum resources, mainly in Alaska and the Stillwater, were estimated to be another 180 to 200 million ounces (Table 2-9).

Within the 1995 time-frame, the market conditions and grade of platinum is probably high enough to warrent production in the Stillwater Complex and from Goodnews Bay. Stillwater ores, at a $400.00 an ounce market level, are probably close to the margin of economic availability. Although platinum prices have been somewhat volatile in recent years, closely following the price of gold, a long, slow, steady inflationary renewal over the next decade or two worldwide would provide a hedge for platinum investors that should drive platinum prices toward the $600.00 to $800.00 an ounce mark. At the same time, the movement in the U.S. and world automobile industry toward increasing production and the continued use of catalytic converters for air pollution control will underpin the platinum demand of the world for the next 15 years.

Duluth Complex Pt ores are not economic or recoverable independent of other metals; and therefore, the platinum production would depend on copper/nickel development. None is expected by 1995. In the Stallwater Complex, with combined mining rates by Anacanda (1000 tpd) and J-M (4500 tpd) of 3500 tpd of ore, production of PGM could reach 1500 to 2000 oz. per day, or 450,000 to 600,000 oz. per day. Assuming Goodnews Bay and other Alaskan placer deposits combined might produce 15,000 to 25,000 oz. per year of PGM under favorable economic conditions, a total U.S. production of PEM of 500,000 to 600,000 oz. per year by 1995 is expectable. This would represent a significant part of U.S. demand; however, given the total projected demand for

Pt by 1995, the U.S. will remain a significant importer of PGM for the rest of this century.

insert in sequence
w. text.

Chapter 8

BIBLIOGRAPHY

Babitzke, H.R., Barsotti, A.F., Coffman, J.S., et al. No date. The Bureau of Mines Availability System: An Update of Information Circular 8654. Washington, D.C.

Bundtzen, T.K., Eakins, G.R., and Conwell, C.N., 1982, Review of Alaska's Mineral Resources. Juneau: Dept. of Natural Resources, Div. of Geological and Geophysical Surveys.

Cameron, E.N., editor, 1973, The Mineral Position of the U.S., 1975-2000. Madison, Wisc: The University of Wisconsin Press.

Charles River Associates, 1982, Survey of the Cobalt Supply and Demand Picture (by B.J. Reddy). Paper presented at National Materials Advisory Board Committee Meeting on Technological Alternatives to Cobalt Conservation. Boston: Charles River Associates.

Council on International Economic Policy, 1974, Special Report: Critical Imported Materials. Washington, D.C.: U.S. Government Printing Office.

Dames & Moore Minerals and Resources Economics Group, 1980, Cobalt, Chromium, and Nickel Supply, Demand, and Strategic Importance. San Francisco: Dames & Moore.

Fischman, L.L., Project Director, 1980, World Mineral Trends and U.S. Supply Problems. Washington, D.C.: Resources for the Future.

Naldrett, A.J., and Duke, J.M., 1980, Platinum Metals in Magmatic Sulfide Ores. Science 208(4451): 1417-1424.

National Academy of Sciences, Board on Mineral and Energy Resources Commission on Natural Resources, 1978, Technological Innovation and Forces for Change in the Mineral Industry. Washington, D.C.: The Academy.

National Materials Advisory Board, Commission on Sociotechnical Systems, 1981a, Assessment of Selected Materials Issues (NMAB-381). Washington, D.C.: National Academy Press.

National Materials Advisory Board, Commission on Sociotechnical Systems, 1976, Manganese Recovery Technology (NMAB-323). Washington, D.C.: National Academy of Sciences.

National Materials Advisory Board, Commission on Sociotechnical Systems, 1981b, Manganese Reserves and Resources of the World and Their Industrial Implications (NMAB-374). Washington, D.C.: National Academy of Sciences.

National Materials Advisory Board, Commission on Sociotechnical Systems, 1980, Supply and Use Patterns for the Platinum-Group Metals (NMB-359). Washington, D.C. National Academy of Sciences.

Office of Technology Assessment, 1982, Cobalt: Policy Options for a Strategic Mineral. Washington, D.C.: U.S. Government Printing Office.

U.S. Bureau of Mines, 1982a, Chromium Aailability--Domestic. A Minerals Availability System Appraisal (IC

U.S. Bureau of Mines, 1982b, Mineral Aailability--Domestic. A Minerals Availability System Appraisal (IC 8889; by C.C. Kilgore and P.R. Thomas). Washington, D.C.: USBM.

U.S. Bureau of Mines, 1982c, Minerals Availability System (MAS) File. Denver: U.S. Department of the Interior.

U.S. Bureau of Mines, 1980a, MILS: The Mineral Industry Location System of the Federal Bureau of Mines (IC 8815; by A.W. Berg and F.V. Carrillo). Washington, D.C.: USBM.

U.S. Bureau of Mines, 1983a, Mineral Commodity Profiles: Platinum-Group Metals. Washinton, D.C.: USBM.

U.S. Bureau of Mines, 1982d, Mineral Commodity Summaries. Washington, D.C.: USBM.

U.S. Bureau of Mines, 1983d, Mineral Commodity Summaries. Washington, D.C.: USBM.

U.S. Bureau of Mines, 1975, Minerals in the U.S. Economy: Ten-Year Supply-Demand Profiles for Mineral and Fuel Commodities. Washington, D.C.: USBM.

U.S. Bureau of Mines, 1979, Minerals in the U.S. Economy: Ten-Year Supply-Demand Profiles for Nonfuel Mineral Commodities (1968-77). Washington, D.C.: USBM.

U.S. Bureau of Mines, 1980b, Supply Analysis Model (SAM): A Minerals Availability System Methodology (IC 8820; by R.L. Davidoff). Washington, D.C.: USBM.

U.S. Department of the Interior, Office of Minerals Policy and Research Analysis, 1979, Developing a Critical Minerals Index: A Pilot Study (by R.L. Adams, B.A. White, J.S. Grichar). Washington, D.C.: U.S. Department of the Interior.

Availability System Appraisal (by G.R. Peterson, D.I. Bleiwas, and P.R. Thomas). Washington, D.C.: U.S. Department of the Interior.

U.S. Geological Survey, 1982, Occurrences of Selected Critical and Strategic Mineral Commodities in Alaska (Open-file report 82-719). Compiled by E.H. Cobb and P.R. St. Aubin. Washington, D.C.: USGS.

U.S. House of Representatives Committee on Science and Technology, Subcommittee on Science, Research and Technology, 1978, A National Policy for Materials: Research and Resources (Interim Report, Serial WWW). Washington, D.C.: U.S. Government Printing Office.

U.S. Senate Committee on Energy and Natural Resources Subcommittee on Energy and Mineral Resources, Subcommittee on Energy and Mineral Resources, 1981, Strategic Minerals and Materials Policy (hearings). Publ. No. 97-14. Washington, D.C.: U.S. Government Printing Office.

Chapter 9

STRATEGIC MINERAL DEVELOPMENT IN THE U.S. FOR THE YEAR 2010

INTRODUCTION

In Chapters 1 through 3, we have reviewed the major world deposits of the metals of interest and briefly discussed the size and grade of the resource base. The major deposits already identified, most of which are producing ores today, will continue to supply the major part of the mineral demand in 2010. We have estimated that platinum will probably be in production from Alaska and the Stillwater Complex by 1995 and thus it is assumed that these areas will be in production in 2010. It is possible that other presently known U.S. deposits may also come on line by 2010, depending upon metal price, political access, and environmental variables; these possibilities are discussed in the scenarios that follow.

In addition, we have shown in Chapters 6 and 7 that it is very unlikely that significant mining or metallurgical breakthroughs in the mining and processing of the low-grade, high-cost, or low-quality ores that generall exist in the United States will affect the expected supply picture. Most improvements that can be introduced in mining and metallurgy by 2010 cannot be exclusively applied to the U.S. resource base and therefore U.S. deposits are unlikely to become substantially more econimic vis' a vis other world ores.

Based on the conclusions presented in the body of this report, few U.S. deposits will be mineable if the prevailing base economic conditions of the year 1980 continue to the year 2010. Thus, the potential for developing most U.S. deposits is contingent upon interruption of foreign supplies or other political considerations. Such considerations will determine whether substantial future mining of the metals of interest will occur even at elevated costs and prices.

Chromite

We make the initial assumption that chromite demand in the year 2010 will be from 75% to 100% greater than the current world demand. This assumption is based on a shift in chromite use, a generally expanding demand for the commodity due to its irreplaceable characteristics in the development of varying steel and other alloy commodities. A business-as-usual scenario for chromium with a growth to twice current demand, suggests, however, that the world suppliers of chromium in 2010 will still be the same suppliers, namely the Bushveld Complex of South Africa, the Great Dike of Zimbabwe, and some of the other major chromium-producing areas of the world where the resource base is large enough to maintain a 25-year supply (described in Chapters 2 and 3).

There is little doubt that the concentration of chromite mining in a business-as-usual scenario in the year 2010 will reside most importantly in the Republic of South Africa and Zimbabwe. Here the lowest-cost chromite available to the world market will continue to be exploited. The capacity and resources to increase production tonnage are readily available both economically and technically for these African ores. In addition, the South Africa and Zimbabwe ores can be competitive anywhere in the world where seaport smelting facilities exist. Under normal relations, Soviet chromites can also reach the world market, and they have enormous potential for increased production over current rates. Their availability, however, will ultimately depend upon Soviet policy initiatives for expanding their international market share for their mineral commodities. There is little reason to think that chromite ores from Brazil, India, the Philippines, Turkey, Finland, and perhaps Albania, cannot continue to play an important role in supplying part of the world demand. In most of these countries chromite could be mined and delivered to American shores for prices, on a 1982 base, that are about one-half that of the most likely American producer--the Stillwater Complex. Under business-as-usual, no U.S. deposits of chromite can be expected to be in production by the year 2010.

Under conditions of a complete foreign cut-off of supplies, the United States would be able to mine chromite from a few sites that could provide a fraction of the annual demand. The demand is estimated to be somewhere between 1.5 to 2 million tons of chromite per year in the year 2010. Chromite resources capable of supplying some small part of that demand are located in the Stillwater Complex of Montana, the disseminated/podiform deposits in northern California, and perhaps the disseminated and podiform chromites that have been recently identified in Alaska. It is also likely that the chromite resource in the Bird River area of Manitoba, Canada, could also be considered an available "domestic" resource. The Stillwater and Manitoba chromites are high-iron, low-chromium grade ores containing 25% or less CR_2O_3. Although production is possible from these three resource areas, prices of chromite concentrate from these sources would probably range from $300 to $400 per ton, in 1982 dollars. This price would probably be more than double the comparable world chromite price, assuming no differential real cost escalation in mining foreign ores.

Rather straightforward mining and beneficiation technologies are available for the exploitation of U.S. chromites, and their composition is suitable for a variety of current uses. Indeed, future breakthroughs in beneficiation and smelting technologies may lead to the mining of lower grade ores than currently anticipated. The resource base, in terms of astimates of Stillwater, Bird River, Alaska, and podiform/disseminate northern California deposits, is known with only a low degree of probability (USBM, IC8895, 1982). The best estimate that can be made, however, is that the resource base is at least in the range of 200 to 300 million tons of chromite, ranging between a high of 25% CR_2O_3 to a low of 1% CR_2O_3. This resource base might be translated into a reserve of 25 to 50 million tons of ore. With an annual U.S. demand of 1.5 to 2 million tons of chromite, and a rapid rate of development, a very optimistic estimate would be that on the order of 50% of demand could be mined by the year 2010. It is highly unlikely, however, that domestic resources of chromite could supply the entire demand for chromite in the year 2010, regardless of incentives.

In summary, the reliance on chromite supply from domestic resources alone would provide a shortfall of anywhere up to 75% of chromite demand in the year 2010. Although a significant supply, perhaps as much as 1 million tons a year, might be available from U.S. resources, that supply would be very high-cost (relative to alternate world supplies) and would be available for only a relatively short period of time. The maintenance of a 1 million ton a year supply would be possible only for a few years, as the California podiform resources, from which a major part of the production would have to come because these resources have the bulk of the potentially recoverable ore, would only be available for an average of 8 to 10 years because of their small deposit tonnage and very low grade. Strategically, therefore, the U.S. will be dependent upon foreign chromites far into the future, as long as chromite remains an important resource in the development of a materials base for the American economy.

NICKEL

By the year 2010, we estimated U.S. nickel demand to be anywhere from 350,000 to 400,000 tons of new metal per year. Based on available world supplies the price of nickel will probably be in the range of $3.50 to $4.00 per pound, in 1981 dollars. In terms of a scenario of business-as-usual, and with the international free trade of nickel expected to continue, it is highly unlikely that any nickel will be produced in the United States in the year 2010; at best, just a small fraction of U.S. demand might be mined under special conditions. Under a scenario allowing the import of all nickel requirements, and with the very large reserve and resource base available throughout the world, particularly in Central and South America and in the western Pacific, it is highly unlikely that U.S. nickel resources would be an important competitor.

Identified world reserves and resources are about 220 million tons of metal (table 2-5). Laterite ores make up about 80% of the total and massive sulfides about 20%. Low-grade nickel ores (0.1% to 0.5% Ni) (such as the nine million metric tons of nickel in the Duluth Complex of northern Minnesota) have not been estimated worldwide and are very large. High-grade nickel deposits of Canada and of Central and South America, particularly Cuba, Brazil, and the Dominican Republic, of the USSR, the huge nickel laterite reserves and resources of New Caledonia, Indonesia, the Philippines, and the ores of Australia, all provide an enormous resource base capable of supplying world nickel demands at relatively low and competitive prices for a long time to come.

The resource base for providing domestic nickel is essentially restricted to two, perhaps three, locales. The first is the nickeliferous laterites of

California and Oregon, where nickel has been mined commercially in the past at Nickel Mountain. However, declining ore-grades, increasing costs, and a diminishing reserve have made the ores at Riddle either uneconomic, or at best only marginally economic, for potential future mining. These deposits are not be likely to compete with the richer and much more extensive Cuban, New Caledonian, Indonesian, or Philippine laterite resources, which could deliver nickel metal to American markets at somewhat lower prices than from northern California and Oregon.

For the business-as-usual scenario we would probably not see the development of the Duluth Complex nickel resources within the time-frame of consideration unless prices rise twofold. However, the future for nickel probably depends more on copper, which will play the major role in determining whether the Duluth nickel resources will be reconsidered for development. Lastly, the Brady Glacier and other S.E. Alaska nickel ores seem to be a most unlikely domestic supply, even under the conditions of rapid price escalation that could bring northern California laterites and the Duluth Complex into production. Given their location, environmental and institutional constraints, and anticipated developmental and mining costs, the Brady Glacier ores would be the least likely domestic resource to be developed within the 2010 time-frame.

The nickel in the lead ores of the east Missouri Lead Belt may be recovered in the business-as-usual scenario. If cobalt is recovered from these ores, nickel will surely add to the byproduct value. The amounts that can be recovered are very small (see Table 8-8) and most of the costs of production will be borne by cobalt recovery. The nickel in the ores, recovered without cobalt, would not be economic under any foreseeable economic forecast. It seems highly unlikely, therefore, that a domestic supply of nickel, under a scenario of business-as-usual, can be developed, unless the price of nickel essentially doubles in real terms.

Under a scenario of cut-off of foreign supplies except from Canada, the year 2010 could see the development of nickel resources in the United States from the nickeliferous laterites of northern California and from the Duluth Complex of Minnesota. Both these areas, along with Sudbury deposits, could supply nearly all of the U.S. nickel requirements, which we estimate to be in the 350,000 to 400,000 metric ton per year range. The U.S. resources could be available in a market structure estimated to be between $4.00 to $5.00 per pound, in 1982 dollars (Table 8-12). These price estimates may be as much as $3 per pound too low, considering the fact that no production is taking place in 1980 domestically despite current prices of about $3.00 per pound.

If the price of nickel escalated substantially, an enormous incentive would exist for Sudbury to expand production wherever possible, perhaps increasing nickel tonnage, from 50% to 100%. Large tonnages could be mined at Duluth, as could some laterites, through relatively low cost, bulk mining technology. Processing is readily available for both ore types, and a coproduct or byproduct flow would come in the form of copper, cobalt and platinum from Duluth and cobalt, and possibly chromite, from the California laterites. In addition, it is our assumption, that the California laterites will be developed under the foreign supply cut-off as a major cobalt source by the year 2010. Platinum in the Duluth Complex would be an important byproduct, and copper would be the coprimary ore, thereby limiting and defining the extent of development in Duluth Complex. The availability of copper in the world market, U.S. demand, and the price of copper would all be factors in determining whether ornot the Duluth Complex is a viable economic operation and, therefore, whether or not nickel could be derived as a coproduct. Discussion of the nickel-bearing lead ores of east Missouri in the business-as-usual scenario is applicable to this scenario as well; nickel recovery depends upon cobalt capture, and it is unlikely that nickel alone could compete economically with other sources in the U.S.

In summary, therefore, we conclude that in a business-as-usual scenario to the year 2010 it is most unlikely that any U.S. nickel resources will be developed primarily for their nickel content. Those deposits that might produce nickel in the 2010 time-frame are the northern California/Oregon laterites, in which cobalt might be a co-primary ore, the Duluth Complex in which copper would be the primary or co-primary ore, along with byproduct platinum and cobalt, and the eastern Missouri Lead Belt, in which cobalt would be the primary metal from the waste products of lead beneficiation.

The supply of nickel in the case of both northern California laterites and the eastern Missouri Lead Belt would be rather small and could not accommodate all of domestic demand. We see, therefore, relatively little domestic nickel production in a business-as-usual scenario, with Canadian and laterite sources remaining cheap and available in the world market.

In a scenario that contemplates a lack of import capability, exclusive of Canada, perhaps as much as 50% of the U.S. demand could be developed domestically. In that case the major nickel source in the United States would undoubtedly be the four potential development blocks in the Duluth Complex (Chapter V). Smaller amounts could come from laterite deposits. However, as discussed in Chapter V, a portion of this resource may not be available due to environmental considerations. Metal mining could take place at an estimated price approximately double the current price of copper and nickel, thereby developing a huge resource in the Duluth Complex that could support a domestic nickel industry for a long period of time. Brady Glacier, Alaska, is the most speculative potential deposit for development. This deposit could provide up to five percent of domestic demand over a 20 year period, but environmental and infrastructure constraints are large.

COBALT

Considering a current world cobalt price of $10.00 to $12.00 a pound, and a spot market price rapidly falling to levels near $3.00 a pound, it is highly unlikely that a business-as-usual scenario to the year 2010 could develop U.S. cobalt resources without subsidies or government purchase agreements. The world cobalt supply picture is dominated by Zaire and Zambia, but cobalt is also potentially available from a variety of other resources, including recovery of significant amounts from the large nickeliferous laterites of the western Pacific. These sources should dominate the free market cobalt supply for the next 30 years. Estimates of U.S. cobalt consumption in the year 2010 reach twice the current rates based upon a 2.8% growth rate.

Although the price of cobalt has vaulted during the past few years, and at one time reached levels of $40.00 a pound, stable and low growth rates of the world market economies, together with advances in technology related to the production and use of alloys, should continue to keep the large traditional cobalt supply in the forefront of the market. Unless the price of cobalt moves to levels of $20.00 a pound, in 1982 dollars, a U.S. supply will not be developed in the business-as-usual scenario to the year 2010. The Blackbird Mine of central Idaho would be the targeted deposit for development at a time when cobalt reached that $20.00 equivalent price level. At $25.00 a pound for cobalt, the Madison Mine can be developed and thereafter, east Missouri Lead Belt ores can be recovered for cobalt, and the nickeliferous laterite ores of California and Oregon might also yield some recovery from the richest of those deposits. Without subsidy, or a purchase agreement contract for the U.S. stockpile, for example, we will probably not see even the best of the U.S. cobalt deposits developed within the next 30 years under a business-as-usual scenario.

The world major supply base of cobalt can support a world increasing demand for the metal for a very long time. Clearly, over the next 25 years, there should be no shortage in identified cobalt reserves and inferred resources that could be put on the market in a relatively short period of time (i.e. within 7 to 10 years) and which can be mined and sold at prices cheaper than any U.S. deposit.

Under a scenario of domestic development caused by exclusion of foreign imports, the U.S. supply of cobalt could be developed from three critical areas of identified cobalt resources. Any or all of these areas could be in production by the year 2010, and the total annual tonnage of cobalt producible from these areas could amount to between 15% to 25% (8.5 million pounds/yr) of the domestic demands in the year 2010. With the addition of the four remaining sites discussed in Chapter V of the Duluty Complex, another five to six million pounds per year might be available.

For U.S. resources, the cobalt ores that can be developed by the year 2010 are the Blackbird Mine in central Idaho; nickeliferous, cobalt-bearing laterite deposits of northern California; and byproduct cobalt of the east Missouri Lead Belt and the Duluth Complex. Perhaps some small cobalt recovery is possible from a variety of other ores in which cobalt is a rather unimportant accessory or trace element (e.g. Brady Glacier). The total amount of cobalt that might be available, in a 2010 scenario of domestic supply from these three primary deposits, could not support the domestic demand for cobalt. Also, within a relatively short period of time, from 10 to 15 years, we would see declining production from these sources, especially from the East Missour Lead Belt, and perhaps Blackbird if new resources are not developed. Therefore, under an extreme situation where foreign cobalt was not available, the U.S. would not be able to support its total cobalt needs at any conceivable or rational domestic price.

If one looks for resources of cobalt that might be available in the price range of $25.00 to $50.00 and $50.00 to $100.00 a pound, it is difficult to visualize or characterize the deposits that would be available. In fact, none are now known beyond what has been discussed that could bring cobalt on line in tonnages large enough to be important for industrial uses in the year 2010. For cobalt, therefore, only substitution, recycle, and intense conservation efforts could help provide us with a steady flow of reduced supply in the year 2010, should foreign resources become unavailable.

PLATINUM

Platinum is notable in that business-as-usual conditions will result in some production in the United States--the platinum placers of Goodnews Bay, Alaska, and the Stillwater Complex of Montana. The potential platinum resources of the United States are modestly large, and include the Stillwater deposits, and extensions of the platinum mineralization, as well as byproduct recovery of platinum from the Duluth Complex, as well as Alaskan placers. The Stillwater platinum deposits are currently marginally economic and could be brought on-line with little or no change in the price of platinum, and with the current institutional and environmental restrictions that will apply to mining in the Stillwater. We have suggested that probably will be in production by 1995. The identified platinum resource base at Stillwater has been estimated to be as large as 4.5 million troy ounces of PGM (U.S.G.S., 1972), however, the recoverable portion might be larger under conditions of relatively high platinum price. Mining in the Stillwater would be underground, relatively large bulk, low-cost operations, and platinum recovery at current market prices would nearly offer reasonable profit margins. Annual production could reach 400,000 to 500,000 ounces of PGM per year, in a total market estimated to reach 4.5 to 6 million troy ounces by 2010. For those companies undertaking development in the Stillwater, work in 1980 involves reserve base identification and defining the economic parameters of production so that mine life can provide the necessary return on investment.

A business-as-usual scenario for the year 2010, however, would provide most of the platinum to the United States from the traditional platinum resources of the world; mainly the Republic of South Africa, the USSR, and a significantly smaller but important amount from Canada. These major world platinum producers capture the low cost, most readily available platinum in the world resource base. A small platinum flow can also be derived from U.S. placer deposits, primarily in Alaska and of the Goodnews Bay type. At the current world price of platinum these placer deposits are now marginally economic to produce, and probably have a cut-off, in 1982 dollars, of approximately $400.00 an ounce. There is no other platinum production than can come on-line in the United States by the year 2010 at the equivalent of the 1982 world platinum price.

At the current price the incentive to explore for new platinum placer reources in Alaska is not very great. On the other hand, a doubling or tripling of the current world price would lead to an exploration effort that should intensify over the next 30 years and could lead to a modest but important platinum mining industry. It is unlikely, however, that the U.S. could come within an order of magnitude of the production tonnage from the Republic of South Africa or the USSR.

In a scenario of no imports, the United States would be hard pressed to develop a platinum supply equal to the industrial demand in the year 2010. That demand is estimated to be in the order of 4.5 to 6 million troy ounces per year of platinum and platinum-group metals. The current U.S. platinum production, in the order of 6,000 troy ounces per year, could not be increased by two orders of magnitude, unless the price of platinum rose by 2 or 3 times in 1982 dollars. The major constraint to such production would be the size of mining operations needed to recover substantial amounts of platinum from the Duluth Complex. In addition, a market for the copper and nickel recovered from the smelting of Duluth ores must be found for any mining to occur.

Under conditions of no imports, and increases in platinum values to 2 or 3 times current price, new Alaskan placers would be discovered and low grade deposits mined by the stimulation of exploration in Alaska, the Stillwater Complex in Montana, would be actually mined and the Duluth Complex of Minnesota could provide a small part (100,000 troy oz/yr.) of the PGM demand for the United States in the year 2010. It is our estimate, however, that at platinum prices of $800.00 to $1,000.00 an ounce, U.S. platinum demand could not be fully accommodated from the three deposits discussed. However, the total amount of platinum that could be mined at substantially elevated prices is difficult to estimate.

MANGANESE

The current manganese resources of the United States, though low grade and of low quality, and in part unusable, nonetheless represents a rather large resource base, situated mainly in Aroostook County and Maple/Hovey Mountain, Maine, and in the north Cuyuna deposit of Minnesota. Smaller amounts of manganese, some of it of higher grade, occurs in Arizona and Colorado as well as in Montana, and makes up a very small portion of the resource base; it cannot be mined as a primary source of manganese under most conceivable economic conditions. In addition, manganese demand through the next 25 years may well be significantly smaller than today, as the steel industry changes metallurgical and product specifications in the development of new steel technologies.

In a business-as-usual scenario for the year 2010 no primary manganese would be mined in the United States. The major world resources of manganese, in particular the USSR, the Republic of South Africa, Gabon and Brazil, together with Australia and India, can readily expand production and meet all world market economy demand in the year 2010. They certainly can supply the United States with manganese, estimated to be in the range of 1.5 to 2 million tons a year by the year 2010 given no changes in manganese use. In a business-as-usual scenario there is no conceivable set of circumstances that could bring primary American manganese deposits on-line to provide anything more than a very, very small part of U.S. consumption.

On the other hand, a no-import scenario would bring us to a very difficult realization with regard to manganese use in the United States. That realization is that supplying the amount of manganese required by the economy would require a five to 10 fold increase in the current price of manganese, in

1981 dollars. (USM, 1982) There may be few limits to the amount of manganese that could come from Maine and Minnesota, provided that beneficiation and some environmental problems are overcome and manganese utilization can be tailored to the grade and quality of the manganese that is produced. The assumed highest grade American sedimentary manganese ores are in the neighborhood of 15% Mn and comprise a resource base large enough to provide at least 10 and perhaps as much as 25 million tons of contained manganese. These ores occur in a geographically small part of the mineralized Chamberlain, South Dakota, deposits and might be mined even before the Maine or Minnesota ores. These low grade nodular manganese ores in soft sedimentary rocks are associated with near surface, relatively low-cost, potentially high-volume mining operations that could well provide a very large amount of manganese in a relatively short lead time. However, continuity of ore grade and tonnage are not well known and the USBM (1982) did not consider these deposits in their U.S. manganese analysis.

In summary, in the year 2010, with a leadtime of 5 to 7 years, we could produce the tonnage of manganese required by the domestic industry from the low-grade ores at relatively high costs of production. Prices 5 to 10 times current ones would make domestic manganese production feasible (USBM, 1982, Figure 3). The total resource base, however, has not been fully explored, and the supply structure is uncertain beyond the year 2000; conceivably more than 10 to 15 years of manganese supply is available from the Chamberlain, South Dakota ores and with the Maine and Minnesota ores supplying the market as well, the domestic deposits could support a large part of U.S. demand for a long time.

SUMMARY AND CONCLUSIONS

In summary, strategic and critical minerals picture for the United States appears as follows for the year 2010. Under a scenario of business-as-usual it is unlikely that American deposits will produce any chromite, manganese, or nickel; only small amounts of platinum and possibly cobalt could be produced. Few domestic deposits will be able to compete economically with materials available on the world market. It is only in a scenario of import restriction or cut-off that the United States could perhaps develope domestic resources for the first three metls, or expand platinum and, to a lesser extent, cobalt.

Specifically, in a non-import scenario for the year 2010,we may be able to produce the low-grade chromite ores of the Stillwater Complex of Montana and the podiform or disseminated deposits of northern California and southern Alaska. We do not anticipate production levels that could meet normal U.S. demand, and important reductions of chromite and ferrochrome use would have to occur to balance supply and demand from domestic resources. In addition, that supply could not come on line without a doubling or 2.5-fold increase in the price, based on 1982 dollars, in the year 2010.

For cobalt, ores could be available in 2010 at the equivalent 1982 market price of $20.00 a pound from Blackbird, Idaho, and at $25.00 a pound from the east Misouri Lead Belt, where both cobalt and nickel would be recovered from the waste sulfides mined with the primary lead-zinc ores. Some small amount of cobalt is recoverable from the nickeliferous laterites in northern California and would owe its market availability to the economics of nickel rather than cobalt. Small amounts of cobalt could also come as a byproduct from copper/nickel mining in the Duluth Complex. In the year 2010, U.S.

supplies of cobalt would not be able to provide the total domestic demand because the resource base outlined above is probably not able to meet the required level of production.

Again, in the year 2010, nickel ores below a price level of $8.00 a pound, in 1982 dollars, would not be produced in the United States. However, at that price, nickeliferous laterites from northern California, the nickel ores of the Duluth Complex, and the nickel byproduct metal of the east Missouri Lead Belt might be available for production. The price of nickel, essentially twice the current price in real terms, could bring a significant amount of nickel to the U.S. market from domestic resources. A total cut-off of foreign supply might add the nickeliferous copper ores of Brady Glacier and other S.E. Alaska deposits to the U.S. supply base, continue the expansion of the Duluth Complex, and bring on the production of the lower-grade nickeliferous laterites of northern California. Essentially, the sources would remain the same, but expanded mining and a larger supply would be available at prices somewhat greater than the $8.00 threshold price, which is the estimate for the Duluth Complex commercialization.

Manganese ores could be available at 5 to 10 times the current price of manganese, primarily from Chamberlain, South Dakota, and the deposits of Aroostook County and Maple/Havey Mountain, Maine, and Cuyuna Range of Minnesota. These latter ores might pose some beneficiation and metallurgical application problems, which could easily be resolved in the nearly 30 years to the 2010 benchmark year. Under a business-as-usual scenario, no manganese will be mined in the United States in 2010 and no significant amount of manganese would probably be mined in the United States until long after that year.

Platinum ores will be provided by the Stillwater Complex, and the richer placer deposits of Alaska, from now until the year 2010. These ores are clearly available as part of the U.S. resource base and could provide a small amount, perhaps up to 5% to 10%, of U.S. domestic needs in the year 2010. A critical cut-off of platinum supply on a worldwide basis would lead to the development of platinum from the Duluth Complex and the expansion of placer mining in Alaska. Platinum in the Duluth Complex would only be recovered along with nickel and copper and, therefore, the primary metals would control the availability of platinum. It is conceivable that total domestic platinum needs could, for a short period of time, be supplied by the mining of the Duluth Complex, along with the known placer resources and the Stillwater. It would rquire, however, that the Duluth Complex produce from 3 to 5 million troy ounces per year, in a great mining enterprise as yet unequaled in the United States. The constraints to an operation of that size are many, especially in terms of environmental and institutional limits and alternate use of the resource area. It is doubtful, therefore, that the United States will ever be self-sufficient in platinum, regardless of expected or forecasted upper price limits.

Chapter 9

BIBLIOGRAPHY

Babitzke, H.R., Barsotti, A.F., Coffman, J.S., et al. No date. The Bureau of Mines Availability System: An Update of Information Circular 8654. Washington, D.C.

Bundtzen, T.K., Eakins, G.R., and Conwell, C.N., 1982, Review of Alaska's Mineral Resources. Juneau: Dept. of Natural Resources, Div. of Geological and Geophysical Surveys.

Cameron, E.N., editor, 1973, The Mineral Position of the U.S., 1975-2000. Madison, Wisc: The University of Wisconsin Press.

Charles River Associates, 1982, Survey of the Cobalt Supply and Demand Picture (by B.J. Reddy). Paper presented at National Materials Advisory Board Committee Meeting on Technological Alternatives to Cobalt Conservation. Boston: Charles River Associates.

Council on International Economic Policy, 1974, Special Report: Critical Imported Materials. Washington, D.C.: U.S. Government Printing Office.

Dames & Moore Minerals and Resources Economics Group, 1980, Cobalt, Chromium, and Nickel Supply, Demand, and Strategic Importance. San Francisco: Dames & Moore.

Fischman, L.L., Project Director, 1980, World Mineral Trends and U.S. Supply Problems. Washington, D.C.: Resources for the Future.

Naldrett, A.J., and Duke, J.M., 1980, Platinum Metals in Magmatic Sulfide Ores. Science 208(4451): 1417-1424.

National Academy of Sciences, Board on Mineral and Energy Resources Commission on Natural Resources, 1978, Technological Innovation and Forces for Change in the Mineral Industry. Washington, D.C.: The Academy.

National Materials Advisory Board, Commission on Sociotechnical Systems, 1981a, Assessment of Selected Materials Issues (NMAB-381). Washington, D.C.: National Academy Press.

National Materials Advisory Board, Commission on Sociotechnical Systems, 1976, Manganese Recovery Technology (NMAB-323). Washington, D.C.: National Academy of Sciences.

National Materials Advisory Board, Commission on Sociotechnical Systems, 1981b, Manganese Reserves and Resources of the World and Their Industrial Implications (NMAB-374). Washington, D.C.: National Academy of Sciences.

National Materials Advisory Board, Commission on Sociotechnical Systems, 1980, Supply and Use Patterns for the Platinum-Group Metals (NMB-359). Washington, D.C. National Academy of Sciences.

Office of Technology Assessment, 1982, Cobalt: Policy Options for a Strategic Mineral. Washington, D.C.: U.S. Government Printing Office.

U.S. Bureau of Mines, 1982a, Chromium Aailability--Domestic. A Minerals Availability System Appraisal (IC

U.S. Bureau of Mines, 1982b, Mineral Aailability--Domestic. A Minerals Availability System Appraisal (IC 8889; by C.C. Kilgore and P.R. Thomas). Washington, D.C.: USBM.

U.S. Bureau of Mines, 1982c, Minerals Availability System (MAS) File. Denver: U.S. Department of the Interior.

U.S. Bureau of Mines, 1980a, MILS: The Mineral Industry Location System of the Federal Bureau of Mines (IC 8815; by A.W. Berg and F.V. Carrillo). Washington, D.C.: USBM.

U.S. Bureau of Mines, 1983a, Mineral Commodity Profiles: Platinum-Group Metals. Washinton, D.C.: USBM.

U.S. Bureau of Mines, 1982d, Mineral Commodity Summaries. Washington, D.C.: USBM.

U.S. Bureau of Mines, 1983d, Mineral Commodity Summaries. Washington, D.C.: USBM.

U.S. Bureau of Mines, 1975, Minerals in the U.S. Economy: Ten-Year Supply-Demand Profiles for Mineral and Fuel Commodities. Washington, D.C.: USBM.

U.S. Bureau of Mines, 1979, Minerals in the U.S. Economy: Ten-Year Supply-Demand Profiles for Nonfuel Mineral Commodities (1968-77). Washington, D.C.: USBM.

U.S. Bureau of Mines, 1980b, Supply Analysis Model (SAM): A Minerals Availability System Methodology (IC 8820; by R.L. Davidoff). Washington, D.C.: USBM.

U.S. Department of the Interior, Office of Minerals Policy and Research Analysis, 1979, Developing a Critical Minerals Index: A Pilot Study (by R.L. Adams, B.A. White, J.S. Grichar). Washington, D.C.: U.S. Department of the Interior.

Availability System Appraisal (by G.R. Peterson, D.I. Bleiwas, and P.R. Thomas). Washington, D.C.: U.S. Department of the Interior.

U.S. Geological Survey, 1982, Occurrences of Selected Critical and Strategic Mineral Commodities in Alaska (Open-file report 82-719). Compiled by E.H. Cobb and P.R. St. Aubin. Washington, D.C.: USGS.

U.S. House of Representatives Committee on Science and Technology, Subcommittee on Science, Research and Technology, 1978, A National Policy for Materials: Research and Resources (Interim Report, Serial WWW). Washington, D.C.: U.S. Government Printing Office.

U.S. Senate Committee on Energy and Natural Resources Subcommittee on Energy and Mineral Resources, Subcommittee on Energy and Mineral Resources, 1981, Strategic Minerals and Materials Policy (hearings). Publ. No. 97-14. Washington, D.C.: U.S. Government Printing Office.